LONGWALL DEMONSTRATION IN WESTERN CANADIAN COALS

PHASE 1 - MINE PLANNING & SYSTEMS ENGINEERING

CONTRACT #OSQ82-00107

VOLUME 4

APPENDICES - LONGWALL EQUIPMENT

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Prepared by

McIntyre Mines Limited

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APPENDIX 4-1

TENDER DOCUMENT - THE SUPPLY OF A COMPLETE LONGWALL MINING SYSTEM AND SUPPORTING SERVICES

THE SUPPLY OF A COMPLETE LONGWALL MINING SYSTEM AND SUPPORTING SERVICES

AUGUST 1982

THE SUPPLY OF A COMPLETE LONGWALL MINING SYSTEM AND SUPPORTING SERVICES

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THE SUPPLY OF A COMPLETE LONGWALL MINING SYSTEM AND SUPPORTING SERVICES

1.0 GENERAL INFORMATION

1.1 Introduction

Tenders for the supply of a complete longwall mining system and supporting services are sought by:

McIntyre Mines Limited
14th Floor, Three Calgary Place
355 Fourth Avenue S.W.
Calgary, Alberta, Canada
T2P 0J3

The equipment will be installed at the No. 9G Mine at McIntyre's Smoky River operation located approximately 225 miles slightly north of west of Edmonton and approximately 110 miles by road northwest of the town of Hinton in the Province of Alberta. See Drawing No. 1.

1.2 Access

A paved all weather highway extends from the Edmonton to Jasper highway to the minesite (Highway 40). Rail access is provided by the Alberta Resources Railway which connects to the main national rail system at Jasper.

1.3 Scope of Supply

It is McIntyre's intention to solicit tenders for the supply of a complete longwall mining system to be installed and set to work at its 9G Mine at Smoky River to extract a section of the approximately 20 foot thick No. 4 Seam by the retreat mining method. It is envisaged that the support systems will operate on a coal floor and contact the strata immediately above the coal seam.

The tenders should include but may not be limited to:

Support systems for the maingate, tailgate and face

Coal face conveying system

Coal mining machine or machines

A stage loader and coal transfer system which will discharge the coal mined to a conveyor belt to be installed by McIntyre

A stage loader and coal face lighting system

A stage loader and coal face signalling system with prestart alarm and lockout facility

All electrical equipment associated with the longwall installation including motors, cables, cable terminations switchgear and all power transformers, whose primary voltage shall be 6.6 kv, etc.

All mechanical equipment associated with the longwall installation including gear boxes, drive units, haulages, etc.

All hydraulic equipment associated with the longwall installation including pumps, tanks, fluids, valves, hoses and hose connections, etc.

The equipment to be provided must be new, unused and of modern design consistent with the current levels of technology. Equipment proposed must be suitably approved and acceptable to Alberta Regulatory Authorities.

Normally in retreat mining systems, to ensure that voltage drop and hydraulic head losses are minimized, power transformers and hydraulic pumping systems are maintained adjacent to the face line and retreated with the longwall equipment together with face switchgear. The difficulties that arise when such large items of equipment are to be routinely moved require that this problem be fully addressed in any bid submitted.

McIntyre wish to consider the application of modern techniques which will allow both hydraulic pumping stations and power supply transformers to be located remote from the face and request that the cost of providing these additional features be separately identified in the bid submitted.

It is unlikely that one manufacturer will be able to supply the total range of equipment required to provide the complete installation. McIntyre therefore require that the physical compatability of the equipment proposed be demonstrated before the equipment is dispatched for installation in the mine. In providing such a

compatibility trial, due consideration must be given to the varying gradients likely to be encountered on the coal face and the relationship between coal face and gate road equipment.

Any costs that may be charged to McIntyre for such a trial shall be identified separately in the tender document.

Since the longwall installation proposed will be unique to Western Canada, the tenderer will be required to provide an adequate number of persons with the necessary technical expertise to assist McIntyre during installation and operation of the longwall face.

McIntyre recognizes that to ensure successful operation of the longwall, adequately trained management, operatives and craftsmen are necessary. The tenderer should therefore provide a recommended training schedule and cost for each category. The costs for provision of technical assistance during installation and operation and the provision of training should be identified separately in the tender document.

1.4 Conditions of Supply

The equipment offered must, as a minimum, comply with the technical specification provided. However, this does not preclude the supplier from quoting for any arrangement differing in part from the specification, provided satisfactory reasons for such a departure are given.

The supplier shall offer a complete package of equipment capable of safe and efficient operation.

The equipment must comply with the conditions of final acceptance and guarantee set out in this document.

All equipment and accessories must comply with the appropriate standards of the Occupational Health and Safety Division and the Coal Mines Safety Act and Regulations of Alberta.

All materials supplied shall conform to C.S.A. standards or its equivalent. Where an equivalent standard is employed, it shall be specified by the supplier.

The proposed Terms and Conditions for Purchase are provided in Appendix 1.

In order to expedite the submission of a tender document, it is suggested that equipment which has received certification in its country of origin be specified and the requisite approval be identified.

In the event that an order be placed with a supplier, it shall be the suppliers obligation to obtain the necessary Alberta approvals. Any costs which may be incurred to obtain such approvals shall be borne by the supplier.

1.5 Panel No. 1 Longwall Face Specification (Summary)

GENERAL INFORMATION	COMMENT
Objective	
1. Available Working Time	390 Minutes
2. Shears per shift	3.4
3. Tons per shift	1 343
4. Tons per day	4 029
Special Considerations	
1. Ventilation/Methane Drainage	Adequate air quantity. CH_4 not expected to be a problem. Fit methane monitor on power loader.
2. Spontaneous Combustion	Not a problem, but leave prepared sites.
3. Heat and Humidity	No problem.
4. Respirable Dust	Regulations: Static Sample 5 mg/m^3. Coal will be dusty.
5. Materials Transport	Manufacturer's recommendations for support transport.
6. Water	Small quantities of roof water during development and working.
7. Pumping	Required to ensure containment of roof water.
8. Incendive Temperature Potential	Cutting in seam; not high. Roof strata; moderate to high.

GENERAL INFORMATION	COMMENT
1. Date of Start of Production	November 1983
2. Date of Equipment Required	July 1983
3. Method of Working	Retreat
4. Length of Face	578 feet centre to centre.
5. Gradient of Face	Maximum 21° - 4°
6. Length of Main Gate 525' Plan	5200 feet
7. Length of Panel Run	5000 feet
8. Gradient of Main Gate 320'	4°
9. Hand of Face	Right
10. Seam Thickness	20 feet
11. Extraction	10 feet, 8 feet at Main Gate.
12. Depth of Web	2 feet, 3 inches. Double ended ranging drum shearer.
13. Power Loader and Ancillaries (prestart warning - sound/water or both)	Hydraulic, chainless haulage. Speed up to 28 feet per minute. Drum speed order of 20 R.P.M. Powered cowl - suppression inbuilt. Two speed gear box for drum. Solid underframe. HSV.PFF facility. Lump breaker.
14. Face End Arrangement - Main Gate	AFC delivers from step onto stage loader. Road head with two supports which may move stage loader.
15. Face End Arrangement - Tail Gate	AFC return drive in tailgate. Road head with one support.
16. A.F.C. Main Drive	Manufacturer's recommendations.

GENERAL INFORMATION	COMMENT
17. A.F.C. Tail Drive	Manufacturer's recommendations
18. Line Pans (including inspection pans)	Manufacturer's recommendations.
19. Chain	26 mm twin strand.
20. Chain Tensioners	Manufacturer's recommendations.
21. Goaf Side Furnishings	Outboard tubular trapping.
22. Cable Handling	Bretby type (plastic) preferred.
23. Face Side Furnishings	Ramp plates with cutaways.
24. Main Gate Anchorage	Face line support or as recommended.
25. Tail Gate Anchorage	Face line support or as recommended.
26. Stage Loader	Beam type onto belt conveyor. Capacity greater than AFC.
27. Powered Supports	Suitable for extraction of seam section from 8 feet to 12 feet.
28. Main Gate Belt Conveyor	Varitrough structure supported from roof 42" wide belt 600 f.p.m. belt speed. Automatic take up required. Must be compatible with stage loader.
29. Lump Breaker (Stage Loader)	Reduce coal to -4".
30. Hydraulic Power Pack (Roof Supports)	Remote station outbye the face preferred.
31. Tank	Manufacturer's recommendations.
32. Hydraulic Props	Length 5'6" to 14'. Power set.
33. Roof Bars	Provided by McIntyre.

GENERAL INFORMATION	COMMENT
34. Shearer Drum	Three start. HSV/PFF. Conical back face. Hoseless. Adequate to completely cut out for gearhead move over.
35. Water Hose	Not less than 1 1/4 inches diameter.

GENERAL INFORMATION	COMMENT
Electrical	
1. Electrical Voltage	1100/1000
2. Transformers	Supplied by McIntyre. Required capacity to be supplied. Manufacturer supply for remote power centre.
3. Switches 11 T Circuit Breaker	Supplied by McIntyre.
4. Tramming Cables	Supplier's recommendations.
5. Trailing Cable for Shearer	Supplier's recommendations.
6. Trailing Cable for AFC	Supplier's recommendations.
7. Switches - Power Loader 8. Switches - AFC 9. Switches - Other	Where feasible, switchgear type and capacity to be standardized. Equipment with monitoring facilities WECOL preferred.
10. Remote Control Switches	
11. Drill Cable	Supplier's recommendations.
12. Drill Switch	Supplier's recommendations.
13. Section Switches	Supplier's recommendations.
14. Face Signals and Communication	Prestart warning for all conveyors. Broadcast communication link to automatic mine telephone system. Davis Derby preferred.
15. Stage Loader Signals	Same as above.
16. Gate Conveyor Signals	Same as above.
17. Electric Rotary Drilling Machine	Victor Products preferred.
18. Stage Loader/Switch Carrier	Supplier's recommendations.

2.0 DESIGN PARAMETERS AND DATA

2.1 Production Requirements

It is envisaged that the longwall will be procured, installed and operated as a cooperative venture between Canmet and McIntyre to demonstrate the potential for the successful application of such a system in Rocky Mountain mining conditions. Although the opportunity will be taken to conduct research projects to provide meaningful data for the design of future installations, it is of paramount importance to rapidly achieve and consistently maintain high levels of productivity. As a minimum, it is anticipated that the proposed installation will provide in excess of 4000 LTR per day when operated on a three shift basis, extracting the top 10 feet of the 20-24 foot thick No. 4 seam. The following data is provided as a basis for design and supply of the required longwall equipment.

2.2 Mining Plan

For the purpose of the longwall demonstration project, two longwall faces have been prescribed. See Drawing No. 2.

The site will provide experience of mining at varying grades and of installation, withdrawal and reinstallation of equipment. Additional faces to those prescribed will be available upon completion of the demonstration project.

2.3 Face Parameters

The faces will be developed from roadways spaced at 550 feet centre to centre in plan with the travel of the first panel being 5000 feet.

The general gradient along the face is in favour of the load and will vary from 18° at the start line to 5° in the later stages of operation of Panel 1. Local gradients of up to 25° on the face line must be accomodated.

Initial equipment is to be supplied for Panel 1 and must be sufficient to equip the longwall face at its steepest gradient. See Drawing No. 2.

2.4 District Roadways

The configuration and dimensions of the roads which will be developed are shown in Drawing Nos. 3 & 4.

The general gradient along the conveyor roadway in Panels 1 and 2 will be 4° against the load.

2.5 Mining System

A system of retreat mining will be employed to extract coal between two sets of pre-driven entries. The faces will retreat in

the direction of reducing cover, ie. for Panel 1 from 1000 feet to 500 feet.

The main gate entry arrangement as shown in Drawing No. 3 will require a lead in extraction height at the maingate end of approximately eight feet.

During the demonstration, the extracted height along the face will be varied between eight and twelve feet. Every effort will be made to maximize recovery of the resource, although the final extracted height will be influenced by the bearing capacity of the various horizons in the coal seam. The system proposed must therefore be capable of operating within this range without modification.

2.6 A.F.C. Stage Loader Configuration

In the main gate entry the floor is shown stepped. With this configuration, the AFC cannot be mechanically connected to the stage loader to ensure consistent coal delivery.

To maintain a proper discharge from the AFC to stage loader a means of steering the stage loader toward the step must be provided, by the supplier.

Variations in face length caused by local changes in gradient may be accommodated by slightly angling the face to maintain the

relative position of the face conveyor gate drive and discharge arrangements.

The supplier shall propose a suitable system and specify the equipment necessary to ensure that a proper transfer of coal can be effected and that the stage loader, conveyor tail end and all other equipment, including pumps, tanks, switchgear and transformers associated with the operation and located adjacent to the coal face can be retracted in an efficient, safe and timely manner.

2.7 Geotechnical and Geological Data

To assist the tenderers in the design and specification of equipment, the following pertinent data is provided as Appendix 2.

1. The results of a program conducted by CMRC to determine the floor strengths at various horizons in the No. 4 coal seam in the 9G Mine.

2. Geological data derived from exploration boreholes in the 9G Mine area.

3. Geotechnical data and observations derived from the adjacent Mine 9A which is extracting the top portion of No. 4 seam by room and pillar methods.

McIntyre is currently conducting a program to derive core samples from the immediate roof and floor of No. 4 seam in the No. 9G Mine. The samples will be sent to Seegmuller International for testing and analysis and the results of this analysis will be made available upon completion of the program.

2.8 Power Supply System

The main power supply system used for distribution in the underground mine is 6.6 kv, 3 phase 60 Hz and has adequate capacity to accommodate the longwall installation.

2.9 Water Supply

A clean water supply capable of providing up to 90 g.p.m. will be available. The water will be obtained from wells drilled from the surface and from water within the mine. The average pH value of the water is expected to be 7.

2.10 Ventilation

Anti-tropal ventilation will be practised with an air quantity of the order of 40,000 c.f.m. passing along the face.

3.0 EQUIPMENT SPECIFICATION

3.1 General

The equipment proposed in response to this enquiry must be certified for use in similar conditions to those likely to be encountered at Smoky River.

Details of the equipment approvals obtained must be supplied with the tender document.

A list of drawings (Table B) required to be submitted is appended at the end of this section together with Summary technical data sheets (Table A) for each item of equipment. This information must be supplied with the tender document. Additional drawings and technical data may be supplied if it is considered appropriate.

If the supplier wishes to propose any alternative or additional equipment for consideration similar supporting data to that requested must be provided.

In order to maximize the potential for success of this mining demonstration, McIntyre request that the most robust equipment consistent with the anticipated duty be offered and require that any equipment proposed be capable of mining and transporting at least 2 million long tons before requiring overhaul or replacement.

3.2 The Shearer

3.2.1 Configuration

The shearer shall be of the double ended ranging drum type, capable of bi-directional cutting equipped with a suitable lump breaker.

The haulage system shall be of the chainless type.

3.2.2 Operation Requirements

The shearer must be capable of extracting a 12 feet section of the coal seam and of cutting a minimum of 6 inches below the base of the armoured face conveyor (AFC).

Cutting speeds of up to 28 feet per minute should be available.

The shearer shall be capable of shearing into the entry at each end of the face to ensure no coal fillets remain to impede the advance of the face conveyor drives and attachments.

The shearer shall be capable of negotiating the maximum changes in gradient likely to be encountered during normal operations.

Clearance under the shearer shall be the maximum practicable within the constraints imposed by the roof beams and the top of the machine. Particular attention must be given to the provision of such clearances where the machine traverses the AFC drive frames.

Two speed drum rotation shall be provided. The selection of drum speed shall be effected by means of a speed selection device and appropriate interlocks shall be included.

The machine shall be capable of providing hollow shaft ventilation and pick face flushing at the cutting drums.

3.2.3 Drums and Cowls

The minimum diameter drums capable of compliance with the performance requirements should be supplied.

The drums shall be capable of cutting an effective web depth of 27".

The drums shall be of the three start helical vane type with wide but adequate pick spacing. Heavy duty picks shall be provided.

Pick face flushing and hollow shaft ventilation systems are required.

All wearing surfaces on the drums shall be hard faced and the method used to achieve this shall be specified, by the supplier.

The attachment of the drums to the shearer should be such that the drums can be readily and safely installed and removed.

Power operated cowls shall be provided.

Cowl support arms shall be designed to assure minimum interference of coal flow to the conveyor.

Wiper seals shall be fitted to the leading edges at the top, bottom and face side of each cowl to minimize dust dispersion.

3.2.4 Dust Suppression

Adequate dust suppression means shall be provided to ensure that dust concentrations can be reduced to a level acceptable to the Occupational Health and Safety Division of Alberta, which require less than 5 mg/m^3 in the general air body.

In addition to the pick face flushing and hollow shaft ventilation systems, cowls must be fitted with dust suppression sprays. A water spray valve switching device shall be provided which automatically ensures that when a

cowl is rotated water is diverted to an appropriate bank of sprays.

If it is considered necessary, in order to meet the dust control standards, additional dust suppression equipment such as underframe sprays or air stream deflectors shall be specified.

The dust suppression water system must be designed to ensure that water is flowing before cutting action can be initiated.

The dust suppression methods proposed should be supported by evidence of adequate performance of similar systems in current operation.

3.2.5 Lump Breaker

A lump breaker of sufficient capacity to reduce mineral to a suitable size to pass beneath the shearer underframe shall be supplied. The type of material that could be encountered may range from coal through mudstone to fine grained siltstone and sandstone.

The breaker shall be fitted ahead of the shearer when the shearer is travelling toward the return gate.

The breaker shall be capable of ranging and equipped with positive stop limits to ensure that damage to the AFC or

roof support canopies cannot occur. A means of protecting the unit from excessive impact loading must be provided.

The drive to the lump breaker should be capable of being disengaged when not required.

3.2.6 Machine Underframe

The main underframe shall be of robust construction and adequate strength to maintain its integrity whilst cutting through hard strata and negotiating conveyor undulations associated with disturbance of the coal seam.

The underframe must be a rigid structure and afford support and location of the machine sections consistent with the machine duty. The machine sections must not be subjected to stress, particularly at jointed sections.

Where access is required for maintenance or changing of components subject to wear, this must be readily available.

A captivation system shall be employed to ensure that the machine and underframe maintain their position relative to the AFC and chainless haulage system throughout the face.

This system shall ensure that any forces transmitted to the underframe from the shearing operation are not transmitted directly to the pan line.

Guide shoes shall be spaced as closely as practicable to assist the negotiation of undulations and shall not directly contact the conveyor structure. They shall be so installed that they may be replaced at any position along the face.

The proposed method of captivation shall be fully described by the supplier, and its advantages over other available systems elaborated.

3.2.7 Haulage System

A chainless haulage system shall be employed throughout the total face length.

An automatic speed control system must be provided. The system should be sensitive to cutting load conditions and be capable of varying the haulage speed without operator assistance within a pre-set range determined by the machine operator.

The system shall be designed to provide adequate haulage capacity for the shearer at the steepest gradients likely to be encountered, ie. 30°.

The system shall be capable of withstanding the forces to which it may be subjected whilst cutting through hard strata and be self cleaning.

The driving mechanism shall maintain its integrity and ability to transmit the required haulage forces throughout the range of local gradient variations likely to be encountered on the face.

The system shall include the provision of an automatically/manually applied parking brake.

The system shall be capable of providing adequate braking power to prevent the possibility of inadvertent machine movement with the power off on the maximum gradient likely to be encountered.

The following information shall be provided with the proposal:

An estimate of the roll back distance on a 30° incline if the power supply to the machine is interrupted.

A description of the method used to prevent inadvertent release of the parking brake.

A description of the method by which an authorized person may release the parking brake to allow movement of the machine whilst the power supply is disconnected.

A rationale for the adoption of the particular haulage configuration proposed.

In support of the proposed system, details of active installations where such a system is successfully employed shall be supplied.

3.2.8 Cable and Hose Handling

The cables and hoses required to supply the shearer must be suitably protected from damage. A handling system which does not require constant supervision during normal operations must be specified.

In proposing such a system, due regard must be paid to the likely gradients and local gradient variations that may be encountered along the face.

3.2.9 Control Systems

Operating controls shall be provided at each end of the shearer, located such that the operator can reasonably be

expected to reach them from a position behind the front legs of the roof supports.

The required controls shall be located in such a position that the operator is protected from falling coal when the cutting drums are in the raised position.

Lockout facilities shall be provided to prevent unauthorized operation of the shearer.

An emergency stop shall be provided, capable of activation from any location along the length of the shearer on the goaf side. Operation of the emergency stop shall latch the mechanism in the stop position. Before the shearer can be restarted the stop mechanism must require manual resetting.

A suitable pre-start warning system must be provided.

A methane monitor shall be supplied and installed in a protected location and provide visible indication of 1% methane concentration and a separate and distinct visible or audible warning at 2% methane concentration in the air body at that location.

To assist in horizon control, a system which indicates that the drums are cutting at the correct horizon must be provided.

Although not required as initial equipment, the shearer must be capable of accepting any one or more of the following control systems at a later date:

Radio remote control of all systems,
Coal thickness indication for roof coal,
Automatic steering to a reference horizon adjacent to the roof.

In order to assist fault location in the event of electrical and hydraulic breakdowns an integral fault location and test system must be provided.

3.3 Armoured Face Conveyor

3.3.1 General

The conveyor shall extend the full length of the face and shall terminate at the edge of a step formed by the junction of the face line and the main gate development entry. The tail section shall be connected to the tail gate entry support.

This specification requires the submission of a proposal to supply a face conveyor with a double strand chain configuration and individual items of conveyor furnishings. However, McIntyre wish to take advantage of the latest

technological developments and would be prepared to consider alternative conveyor configurations, provided the advantages to be gained from such alternatives can be elaborated and quantified.

3.3.2 Operating Requirements

The conveyor assembly proposed must have an adequate capacity to maintain coal clearance capability whilst the shearer is operating at its maximum production rate.

The conveyor must be capable of transporting a minimum of 2 million long tons before line sections or drive units require overhaul or replacement.

Conveyor connector arrangements between conveyor sections must be of adequate strength to withstand the proposed duty and be capable of accommodating the horizontal and vertical deflections to be anticipated.

Where rapid wear of components is anticipated, special steels or other methods to improve component life must be employed.

3.3.3 Conveyor Line Pans

Pans shall be of the bottom plated type and facilities for the inspection of the bottom race must be provided.

Both sides of the pan should be reinforced to provide adequate strength for rack bar and ramp plate connection.

Deck plates should be of substantial construction without compromising chain clearances.

Wearing areas on pan ends should be suitably treated or otherwise manufactured to reduce wear problems to a minimum. The method used to achieve this objective shall be specified.

3.3.4 Conveyor Chain

The conveyor chain shall be 26 mm x 92 mm, of twin strand configuration.

The chain shall be suitably treated after manufacture to minimize corrosion.

Flight bar attachment should be such that removal and insertion of flight bars can be readily achieved.

Chain connectors shall have an equivalent strength to the original chain.

Flight bars shall be spaced at appropriate intervals to ensure that the designed carrying capacity of the conveyor can be achieved.

3.3.5 Trappings

A suitable trapping guide shall be provided between the spill plate and conveyor compatible with the shearer underframe configuration. The trapping guide shall be capable of accepting the loads imposed by the shearer weight and coal cutting activity.

The trapping arrangement must be designed to ensure positive captivation of the shearer and engagement of the haulage mechanism. It should be sufficiently flexible to cope with undulations along the face length.

3.3.6 Rack Bars

Rack bars must form a solid and integral part of the AFC assembly. The attachment of the rack bars to the trapping guide and conveyor pans must be adequate to maintain their integrity while subject to severe vibration and impact loading.

Rack bars shall be capable of withstanding any loads imposed upon them whilst mining through hard strata.

3.3.7 Spill Plates

Spill plates must be of substantial construction and of sufficient height to prevent the overflow of mined coal.

Shearer passage along the face must not be impeded and shearer controls must always be accessible.

Where necessary, inspection plates must be incorporated into the spill plates to effect proper access to all parts of the shearer.

To ensure proper alignment of spill plates, buttress plates or other means of adequate strength shall be incorporated into the design, if considered necessary.

Where face support advancing rams are connected to the conveyor structure, the attachment of the spill plates to the conveyor must be capable of withstanding the forces applied when the supports are advanced in tight flushing conditions.

The spill plates provided must be compatible with the proposed cable handling system.

Cruciforms shall be provided in the spill plates to facilitate lifting of the AFC by the powered support.

3.3.8 Ramp Plates

Ramp plates shall be fitted to the face side of the conveyor to provide efficient clean up and adequate support for the face side shearer shoes.

The ramp plates and attachments must be capable of supporting any load imposed by the shearer.

3.3.9 Drive Frames and Drive Units

The total installed conveyor horsepower shall be sufficient to convey the maximum load anticipated with adequate reserve.

Supporting calculations to determine the required horsepower shall be provided.

Driving sprockets shall be fitted with sealed bearing units and have sprocket rings manufactured from high strength steel forgings.

The sprocket teeth shall be machine cut and heat treated.

Drive frames shall be of robust construction and manufactured from high quality steels. Where necessary, wear resistant steel shall be employed.

Drive frames shall be mounted on substantial floor skids. The assembly of drive frame and skid unit should form a solid box construction and provide a low profile consistent with good coal delivery requirements.
To ensure that the assembly can be readily moved all skid edges should be bevelled to reduce floor ploughing effect.

Suitable ramp plates must be provided from the last standard line pan to the remote ends of the conveyor drive units.

Substantial attachment points for roof support connecting bars must be provided. The drive frames shall not flex under any load applied in this manner.

Drive frames shall be capable of accepting the driving units at either side of the frame.

Suitable provisions must be incorporated into the drive frame to allow all necessary cables and hoses to be routed to the goaf side of the AFC.

Chain stripper assemblies shall be capable of rapid replacement without the need for special tools.

Detachable top pieces formed of suitable wear resistant materials shall be intalled to provide the major wear path on curved portions of the drive frame top chain races.

Since the AFC will not be directly connected with the stage loader, consideration must be given to the location of the skid under the gate drive unit to allow for good coal transfer and downhill creep.

Drive units shall be mounted on the goaf side of the drive frames for this installation. However, the units shall be capable of being mounted at either side of the drive frame.

Each drive unit shall be fitted with a hydraulic chain tensioning device as an integral part of the drive unit assembly. An interlock shall be provided to ensure that the main driving motors cannot be energized whilst the chain tensioning device is engaged.

The drive units shall incorporate a means to allow the fluid coupling to be adjusted to obtain the best torque/speed characteristic for the assembly.

The gear box design shall ensure that adequate lubrication of the moving parts can be maintained at operating gradients up to 30° without special precautions.

3.4 Stage Loader

3.4.1 Operational Requirements

As far as practicable, the stage loader should be constructed from the same components as the face conveyor.

To ease problems of installation, the stage loader package, which shall include the conveyor return end, should be of

modular construction, with the modules provided being of appropriate size compatible with underground transportation facilities. The individual modules should be capable of rapid assembly and disassembly.

The carrying capacity of the stage loader should slightly exceed that of the face conveyor.

The stage loader will function as a solid beam conveyor to accept coal from the face conveyor and discharge the coal to a gate trunk belt.

To ensure efficient transfer and minimize spillage on the conveyor systems the stage loader must incorporate a lump breaker.

3.4.2 Tail Assembly

If the supplier proposes to retreat the stage loader by means of hydraulic rams, the forces must not be applied directly to the conveyor return section but to a suitably fabricated frame surrounding the assembly. Such a frame must be capable of transmitting the necessary forces and form an integral part of the loader assembly.

The rams used to retreat the stage loader must be capable of adequate horizontal and vertical articulation.

Since the stage loader will be retreated adjacent to the face entry step, a means of cleaning up the wedge of coal resulting from spillage or spalling must be provided.

Provisions to maintain chain tension shall be placed at the tail end of the stage loader.

3.4.3 Line Pans

Joints between conveyor sections shall have adequate compressive strength to resist any forces applied to move the conveyor.

Limited vertical movement may be accomodated by the use of compression joints within the structure.

Where the conveyor contacts the floor, edges of the conveyor shall be dished or bevelled to ease movement.

The line pans shall be bottom plated throughout with appropriate inspection means provided to examine the bottom chain and race.

3.4.4 Spill Plates and Attachments

Spill plates shall form an integral part of the complete stage loader assembly and provide additional rigidity to the structure.

They shall be of adequate size to ensure that spillage is eliminated. Particular attention should be paid to the sections ahead of the lump breaker where coal may be expected to accumulate, and at the AFC delivery point, to ensure that personnel passing the delivery head of the AFC are protected.

Battering plates shall be provided at the sections of stage loader in line with the AFC delivery.

A suitable means of routing, protection and handling of all necessary cables and hoses required for face and gate operations shall be provided.

The proposed means for cable and hose handling shall be fully described.

3.4.5 Guards and Platforms

Protective guards and covers shall be provided where necessary.

A walkover platform equipped with handrails shall be placed at suitable locations along the structure.

The walkover assembly shall be designed such that it may be readily removed and relocated along the structure. Location and support of the walkover assembly may be

effected by means of lugs strategically located along the structure.

3.4.6 Drive Head Assembly

The drive frame shall be of rigid construction and as far as practicable, similar to that of the AFC.

Adequate articulation in both horizontal and vertical planes shall be provided in the drive head carriage unit connection.

Once installed, the mounting of the drive head to the carriage structure shall retain its integrity at all times.

The system employed to locate the stage loader discharge to the conveyor shall be robust and ensure that the discharge is centralized to ensure good coal transfer.

Adequate clearances shall be provided to prevent the possibility of inadvertent contact between conveyor and stage loader equipment.

3.4.7 Lump Breaker

A lump breaker of suitable design and capacity shall be incorporated as an integral part of the stage loader assembly.

The lump breaker shall be of the vertical drum type, installed in a suitably designed rigid structure. No reaction forces from the crushing action are to be transmitted to the conveyor deck or chain assembly.

Throat adjustment and overload protection are required.

The driving unit shall be compatible with the stage loader and AFC drive units if practicable.

Adequate guarding must be provided.

The transmission unit should incorporate an overrun brake. A panic bar or safety gate, located in a suitable position shall be provided. Operation of the panic bar or safety gate shall de-energize the stage loader and crusher and latch the mechanism in the stop position. Before the equipment can be restarted, the stop mechanism must be mechanically reset.

3.4.8 Belt Conveyor Return End

The conveyor belt shall be 42" wide.

The structure provided shall be of a rigid box type construction capable of providing the necessary support for the stage loader.

Means shall be provided to adjust belt alignment whilst the conveyor is running. The method of adjustment shall be independent of any primary belt retraction device that may be included in the conveyor system.

A heavily crowned return roller shall be provided together with heavy duty impact idlers, closely spaced to support the conveyor belt at the coal transfer point.

A return belt cleaning plough shall be provided, so located that easy and safe access for cleaning of deposited material can be achieved.

3.4.9 Dust Suppression

Appropriate dust suppression means shall be provided at both stage loader and crusher to ensure compliance with relevant statutes.

3.4.10 Emergency Stop Facilities

An emergency stop and lockout facility shall be provided, capable of operation from any location along the stage loader.

Operation of the emergency stop shall latch the mechanism in the stop position.

Before the machine can be restarted, the stop mechanism must be manually reset.

3.5 Belt Conveying System

During development, McIntyre will install a complete 42" conveyor system. The system will employ varitrough type intermediate conveyor structures suspended from roof bolt plates. The supplier is requested to provide a suitable drive unit, delivery head and automatic loop take up arrangements to accomodate the installed structure.

Particular attention should be paid to the method by which the suspended conveyor structure must be integrated with the return end assembly.

The system proposed shall be compatible with longwall operations.

A description of the proposed system and method of operation shall be provided.

3.6 Powered Supports

3.6.1 Operational Parameters

The equipment will be initially installed at a face gradient of 18°. Local variations in grade of up to 25° may occur along the face line.

Equipment shall be capable of operation at grades up to 30°.

The shearer will extract a nominal 10 foot section at a 27" web depth.
The system proposed must be capable of effective operation within the range of 8-12 feet without need for modification.

The support system shall cover the total face length at the maximum gradient and extend across both road heads. Entry configurations are shown in Drawing Nos. 3 and 4.

Suitable supports, which may differ in design to the main face supports, shall be supplied for the road heads.

Entry supports will have their bases set on a horizontal coal floor with the roof beams contacting an inclined roof. The supports shall be capable of accommodating a roof inclination of up to 30° from the horizontal.

Main gate entry supports will operate independently from the face conveyor.

Extended roof beams shall be specified, if required, to compensate for any offset connections to the AFC drive units.

Hoses between entry and adjacent face line supports shall be suitably protected.

Adequate means to prevent toppling and slewing of supports shall be provided.

The system shall be designed to ensure that a definite reference can be established and maintained from a single support or a batch of supports from which the correct attitude and position of all other supports and the face conveyor can be maintained.

Face conveyor creep compensation and correction devices, capable of moving the whole face conveyor system bodily up grade, shall be provided.

The supports shall offer positive resistance to lateral movement, complete roof and waste cover, constant yield characteristics and low floor loading.

The support density shall be adequate for the initial panels proposed and be capable at a later date of operating at depths of cover of up to 2500 feet.

Powered extending cantilevers must be supplied to provide a 15 ton maximum tip load following passage of the shearing machine.

Powered face sprags must be supplied on every support.

Cycle time for advancing and setting of the supports should be a minimum.

3.6.2 Support Bases

Support bases must be of rigid construction and exert a low floor bearing pressure.

The bases shall provide an unobstructed track for the passage of debris.

The bases shall be capable of being fitted with base lifting devices if not supplied as initial equipment.

Cruciforms or similar means of attaching pulling chains to the bases shall be provided.

The leading edge of bases shall be constructed to minimize resistance to movement over ledges which may remain in the floor.

The minimum floor bearing strength acceptable for use of the supports at yield shall be stated.

3.6.3 Support Legs

Each pair of support legs shall be fitted with a pressure gauge so located that it can be readily observed by face personnel.

Legs shall be accommodated within the bases and roof canopies by positive but flexible means and shall be stable when fully extended.

All support legs whether double or single telescopic shall have a constant yield rate and be double acting.

Adequate exterior protection must be provided to the exposed surface of sliding members of the legs. If considered necessary, protective leg stockings should be provided.

The maximum range that can be achieved by the addition of mechanical extensions, without modification to the linkage between base and canopy, should be stated.

3.6.4 Support Canopies

The canopy should be a rigid fabricated structure, suitably strengthened to resist anticipated stress conditions without deformation.

Fixed lifting points shall be provided to facilitate repair of valves, support legs and rams.

Cruciforms suitably located to facilitate uplifting of the AFC pans shall be provided.

Flushing guards shall be provided as an intrinsic part of the rear shield canopy.

Rear canopy stabilizing jacks shall be provided where necessary.

Special consideration shall be given to the design of leg location points on gate and tail entry shields.

Canopies shall be designed to minimize the potential for interference with adjacent shields.

3.6.5 Face Sprags and Powered Cantilevers

Face sprags, when retracted at the lowest end of the support height range, shall allow maximum clearance for the passage of the shearer.

The design of the powered extending cantilever and face sprag assembly shall be adequate to withstand the forces to be experienced during normal operation.

A tip load of 15 tons and maximum roof cover shall be available when the powered cantilever is extended.

3.6.6 Support Advancing Rams

The rams shall be of heavy duty design, double acting and may be reverse mounted.

They shall have sufficient movement potential to accommodate a full 27" advance of the support or face conveyor when all linkage losses have been absorbed.

Adequate horizontal and vertical movement shall be provided.

Attachment to the AFC shall include provision for incremental movement of the face conveyor along the line of face and a means of self centering the ram attachment.

Design of the support shall ensure that personnel are not normally required to step into the ramway within the floor bases.

The ramway shall present the minimum of obstruction to the passage of debris.

3.6.7 Valve and Hose Systems

Valves shall have pilot and high volume flow circuits separated and be of modular construction.

Valve elements shall be corrosion and erosion resistant.

Valves shall be fully capable of meeting the support performance requirements and achieving and maintaining high setting pressures.

Valve groups shall be located such that they can be operated from a safe position, allow free passage for personnel, are accessible for maintenance and protected from mechanical damage.

Simple bi-directional adjacent control shall be provided. The equipment shall be capable of simple modification to batch control.

Operating features shall include:

Independent control of support legs - for four leg supports, the rear legs may be operated simultaneously.

Lock on conveyor advance.

Contact advance.

Hydraulic base lifting.

Positive set.

Lock in neutral.

Incremental control of anti creep equipment.

Unit isolation shall be provided.

The valve system shall be designed such that control of integrated functions, and separate control for each function shall be available.

The valve system must ensure positive control of all functions such that at any selected position of the control lever, only the function or functions selected can operate.

The system shall ensure that in advance, full advance is obtained unless over-ridden.

Advance rams shall be protected against the possibility of overpressure and vacuum build up.

The capacity of the hose supply and return system shall be sufficient to allow for simultaneous operation of three support units.

Adequate filtration shall be provided at each valve bank.

Inter-unit hosing shall be kept to a minimum and a proper means of controlling the hoses between adjacent units whilst advancing shall be provided.

All hoses shall have staple lock connections.

The main supply and return hose circuits should operate as a ring main system or the supply system should be duplicated.

Pressure drop in the supply system should be minimized and full setting pressures must always be available.

3.7 Hydraulic Power Pack

A suitable hydraulic power pack of adequate capacity and pressure to ensure that at least three support units can be operated simultaneously shall be provided.

The equipment shall be provided with at least 50% standby capacity.

Fire resistant fluid shall be used as the hydraulic medium.

The tank shall have adequate capacity to accommodate the total delivery requirements of the longwall and associated hydraulic circuitry.

The system shall include unloading and pressure relief facilities. Appropriate system protection devices shall be included.

Automatic mixing arrangements to ensure emulsion quality must be provided.

The whole assembly should be mounted on a mobile frame of compact design.

As previously requested, consideration should be given to the location of the power pack relative to the face and the most appropriate location and system specified.

A means of indicating the system operating pressure, in addition to the pressure gauges at the pumping unit shall be provided. This indicator shall be placed such that it can be readily observed by the face control operator.

Since the face is intended to be operated on five days per week only, the system proposed must contain provision to accommodate

any additional fluid that may be released into the system by yielding supports during non operating periods.

To allow experimental determination of the most appropriate leg setting pressure during the mining trials, facilities to vary the pressure to the support legs should be provided.

3.8 Support Systems Ahead of Face

Support systems for both entries shall be provided.

The systems proposed shall be compatible with the longwall face system.

It is anticipated that the required support system would employ a number of single, power operated hydraulic props and bars.

A method for the mechanical handling of the components necessary to effect such support shall be provided.

3.9 Electrical Equipment

3.9.1 General

The main power supply the mine is at 25 kV 60 Hz, 3 phase.

At the mine surface the voltage is reduced to 6.6 kV by a 5 MVA oil cooled transformer. A resistor is connected from

the neutral point of the star connected 6.6 kV winding to ground to restrict ground fault currents to 25 amps.

The main distribution voltage in the underground mine is 6.6 kV. This is further transformed by flameproof transformers of various capacities to the normal utilization voltage of 550 v.

The flameproof transformers also incorporate a restriction in the neutral return path to limit ground fault currents to 5 amps.

Flameproof and intrinsically safe equipment is employed exclusively throughout the mine.

Generally, the mining equipment is of American manufacture and switchgear of British manufacture. Cables are usually obtained from Canadian sources.

In order to maximize the availability of electrical equipment, minimize training requirements and reduce stock levels, similar equipment to that in current use would be preferred. Consideration should therefore be given to the supply of equipment by the following manufacturers.

Gate end boxes	Wallacetown
Signalling and communication	Davis of Derby
Lighting and drill units	Victor Products

Flameproof transformers		Brush
H.T. switchgear		Brush
Cable couplers	H.T.	B.I.C.C.
	L.T.	Victor Products

3.9.2 Longwall Equipment

All the equipment proposed for this installation shall be either of flameproof design or intrinsically safe and certified for use in underground coal mines.
Equipment proposed shall not be contained in enclosures or incorporate any external components manufactured from aluminum or alloys of aluminum.

The operating voltage for all longwall and associated equipment shall be 1000 or 1100 volts.

Lighting systems may be either intrinsically safe or 110 volt flameproof.

Signalling, communication and data transmission systems shall be intrinsically safe.

If hand-held tools are required, they shall be supplied from an approved flameproof unit, be fully protected and operate at 110 v, 3 ph.

The following equipment shall be supplied and installed:

A complete and comprehensive signalling and communication system for both longwall face and stage loader. The system should be capable of providing prestart alarms, lockout and signalling facilities in addition to the communication system. A facility for direct connection to the existing automatic telephone system should be provided.

Sufficient lighting transformer capacity, lighting units and cables to provide an adequate level of illumination throughout the longwall face and along the stage loader. Particular attention shall be paid to the illumination of transfer points.

All necessary motors required for operation of the longwall. All motors provided shall have high torque starting capabilities.

All contactor switchgear required for operation of the motors.

Contactors provided shall, as far as practicable, be similarly rated. Each contactor shall be capable of sequenced, local and remote operation and be equipped with suitable transducers to allow monitoring and

recording of contactor closed/open and current flow.

A limited number of contactors should be equipped with a means of monitoring voltage.

All ancilliary equipment necessary to effect the control of the system, ie. limit switches, remote stop/start units, etc.

All cables and cable terminations necessary to effect a complete electrical installation of the longwall face system from the low tension side of the flameproof 6.6 kv/1100 v transformer. Cables shall be adequately sized for the duty and shall as a minimum, be capable of conveying the anticipated full load current continuously without incurring significant voltage losses.

Flameproof transformers will be supplied by McIntyre.

When considering the provison of a remotely located power supply system, the necessary transformers and voltage regulation devices shall be provided.

3.10 Miscellaneous Equipment

All necessary special tools required for the installation and operation of equipment shall be provided.

A list of tools to be provided shall accompany this proposal.

A recommended list of spare parts necessary to maintain the longwall face in operation for a period of two years shall be supplied together with the total cost to be incurred.

If the supplier is prepared to consign certain spare items to McIntyre or to a suitable storage location in Alberta, such spares should also be identified.

A recommended method for the transport, unloading and installation of the system should be submitted with the proposal. Any additional equipment necessary should be specified.

4.0 TECHNICAL ASSISTANCE

Following delivery, a section of the longwall equipment which will include both ends of the conveyor system and road head supports will be erected at the surface of the mine to demonstrate equipment compatibility and allow operators and supervision to obtain 'hands on' experience.

Whilst the equipment is being erected at the mine surface, during installation of the longwall in the mine and for the first six months of operation, the supplier shall provide the services of three skilled and experienced service engineers at the mine. The cost of providing such assistance shall be separately identified in the Tender.

The supplier is required to consider the requirements for operator and supervisor training and recommend suitable means for training of personnel. A cost estimate on a per person basis and total cost of providing such training programs should be provided.

5.0 SUBMISSION OF DATA

5.1 Technical Data

The data sheets attached must be completed and submitted with the tender document. (Table A.)

5.2 Drawings

The drawing requested must be supplied with the tender document. (Table B.)

5.3 Additional Data

Additional data or alternatives to the specification if submitted must be clearly identified and submitted separately.

Details of similar installations to that required by this specification must be provided.

6.0 SUPPLIER PERFORMANCE

6.1 Pricing

The supplier shall complete the pricing document attached. (Table C.) Any additional equipment offered or equipment proposed which differs from this specification shall be clearly identified and separately costed.

The basis for pricing shall be Canadian dollars C.I.F. minesite.

6.2 Delivery

The total system shall be delivered in accordance with the terms of the purchase order, see Appendix 1.

Delivery shall be effected between 6 and 9 months from the date of order.

6.3 Guarantee (Equipment)

The system shall be guaranteed for a period of twelve months from the start of production.

The supplier shall guarantee:

a) The equipment can be safely and efficiently operated in accordance with all relevant regulations

b) The equipment complies with this specification

c) The system proposed complies with good mining practice

d) The equipment is free from patent defect which may arise from faulty design, poor construction or poor materials used in construction.

During the guarantee period, the supplier must be prepared to remedy any defects that may occur in the most expeditious manner, free of all charge to McIntyre. The supplier shall be responsible for any charges incurred in the removal and refitting of any replacement part, the provision of such replacement parts and cost of transportation.

Final acceptance of the system by McIntyre Mines shall be the end of the guarantee period provided the following performance guarantee can be satisfactorily achieved.

6.4 Guarantee (Performance)

The supplier is required to assess the production potential for the equipment when installed at the Smoky River Mine. Based upon the suppliers experience, calculations to support the proposed level of production shall be provided. The supplier will be required to guarantee that the system provided can consistently produce the predicted levels of output provided that:

a) Mining conditions on the longwall face are reasonable and a 10 foot section is being extracted.

b) Gate and tailroad conditions remain stable and do not cause undue mining difficulties.

c) Operators and supervision are suitably trained and experienced.

d) Mining activities are being conducted in accordance with the suppliers recommendations.

e) All ancilliary services to the face such as power supply, water supply, coal clearance, etc. are not interrupted.

Acceptance of the system by McIntyre shall be conditional upon the achievement of the guaranteed production level for a period of not less than twenty consecutive working days. Desirably this should be achieved during the first six months of operation so that maximum benefit can be obtained from the presence of the suppliers technical staff.

7.0 COMMERCIAL

7.1 Validity

The tender shall be valid until February 4, 1983.

7.2 Terms of Payment

The required terms of payment shall be supplied with the tender document.

7.3 Procurement Financing

It is envisaged that the longwall will be procured, installed and operated as a cooperative venture between McIntyre and Canmet to demonstrate the potential for the successful application of such a system to Western Canadian Rocky Mountain mining conditions. Operation of the longwall will afford the opportunity to conduct meaningful research and derive data upon which future longwall installations can be specified.

Although the primary consideration in final equipment selection will be compliance with the technical specification, in order to maximize the benefit of such a demonstration and ease procurement problems, it is necessary to minimize capital expenditures, particularly initial capital outlay.

McIntyre request that in addition to the detailed cost proposal for the supply of equipment and support services, where practicable, an alternative method or methods by which the necessary equipment and services may be procured be submitted for consideration.

McIntyre may be prepared to consider innovative financing systems.

8.0 TENDERS

8.1 Submission

Three copies of the tender documents shall be submitted for the attention of Mr. Howard D. Owen, Manager of Engineering at the address below:

McIntyre Mines Limited
14th Floor, Three Calgary Place
355 - Fourth Avenue, S.W.
Calgary, Alberta
T2P 0J3

Tenders shall be submitted in two sections: the first section shall contain the technical specification and the second section cost tabulations and procurement financing proposals. Sealed tenders shall be submitted by 12 Noon on Friday, September 17, 1982. Tenders received after this date will not be considered.

8.2 Assessment

Tenders shall be assessed on the following basis:

a) Compliance with the specification

b) The supplier's demonstrated ability to provide the necessary equipment and the background experience and

successful operation of similar equipment installed at other locations.

c) The guaranteed level of performance to be achieved.

d) The capital cost of the equipment.

e) The procurement and financing arrangements proposed.

McIntyre reserves the right to select the most appropriate tender which may not necessarily be the lowest cost.

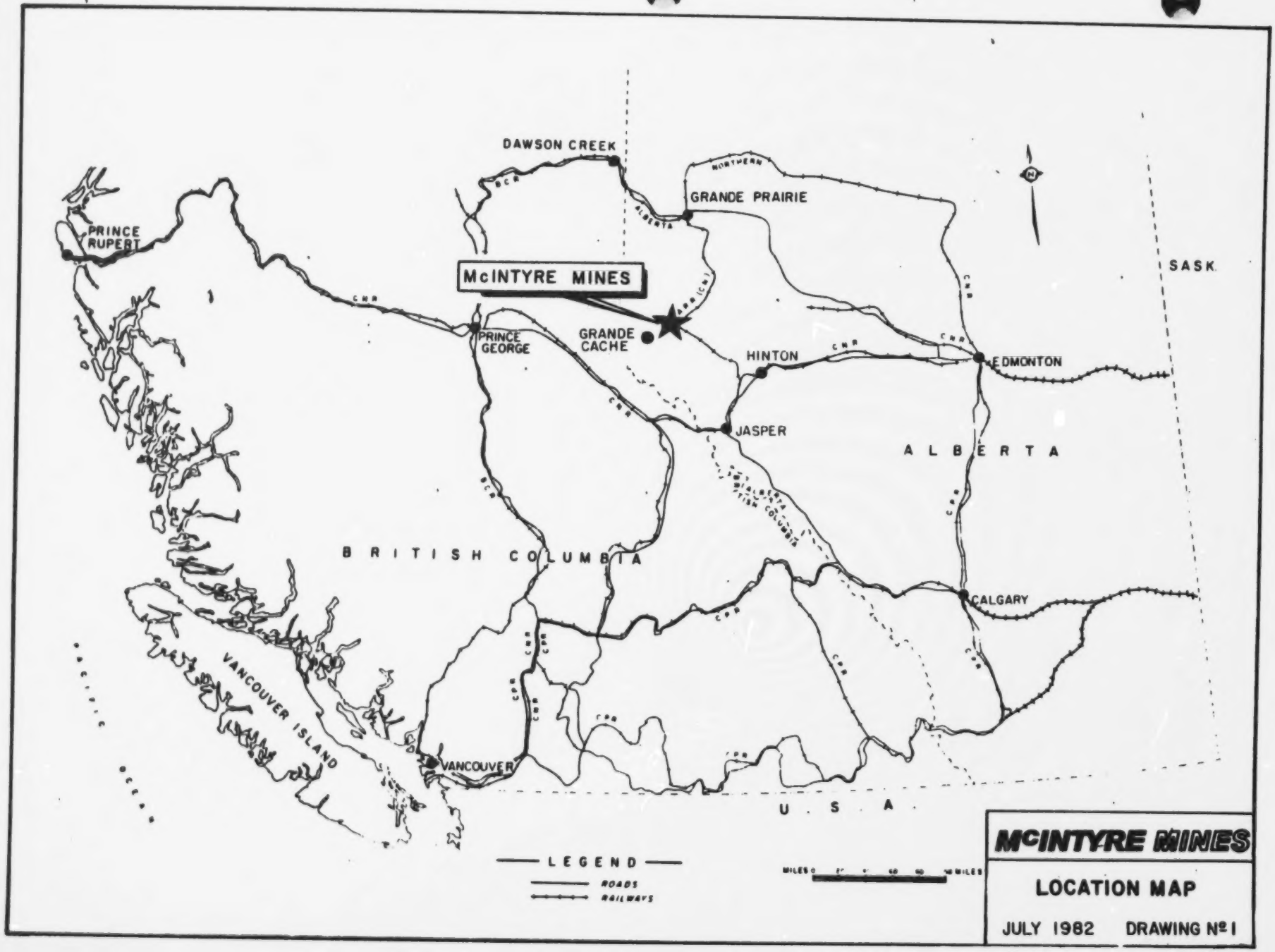
8.3 Notification of Supplier

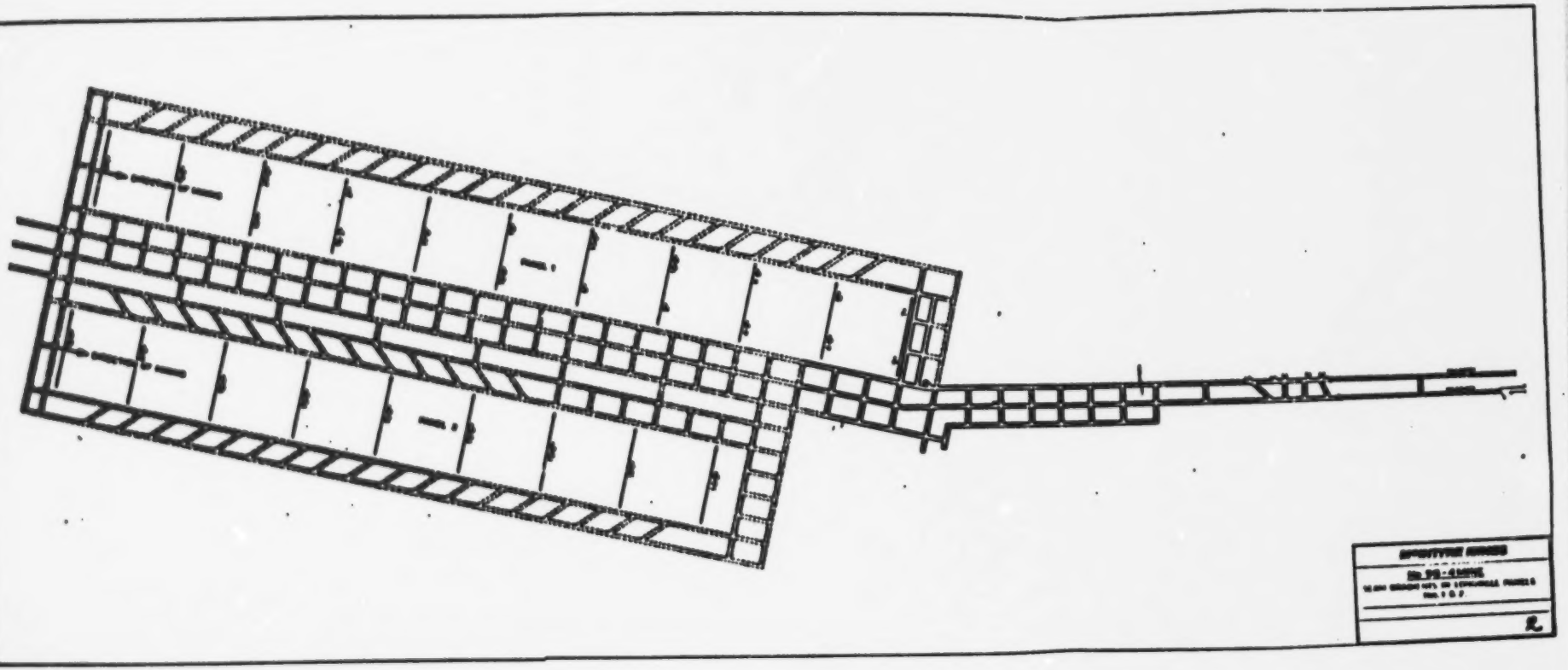
Following assessment of the tenders, McIntyre will inform each supplier of its final choice. McIntyre shall not be required to provide any explanation for the final selection of supplier.

9.0 CLARIFICATION

Should the supplier require any further clarification of any of the items listed in this enquiry document, the request for clarification should be addressed to:

Mr. Howard D. Owen, P. Eng.
McINTYRE MINES LIMITED
14th Floor, Three Calgary Place
355 - Fourth Avenue, S.W.
Calgary, Alberta
T2P 0J3

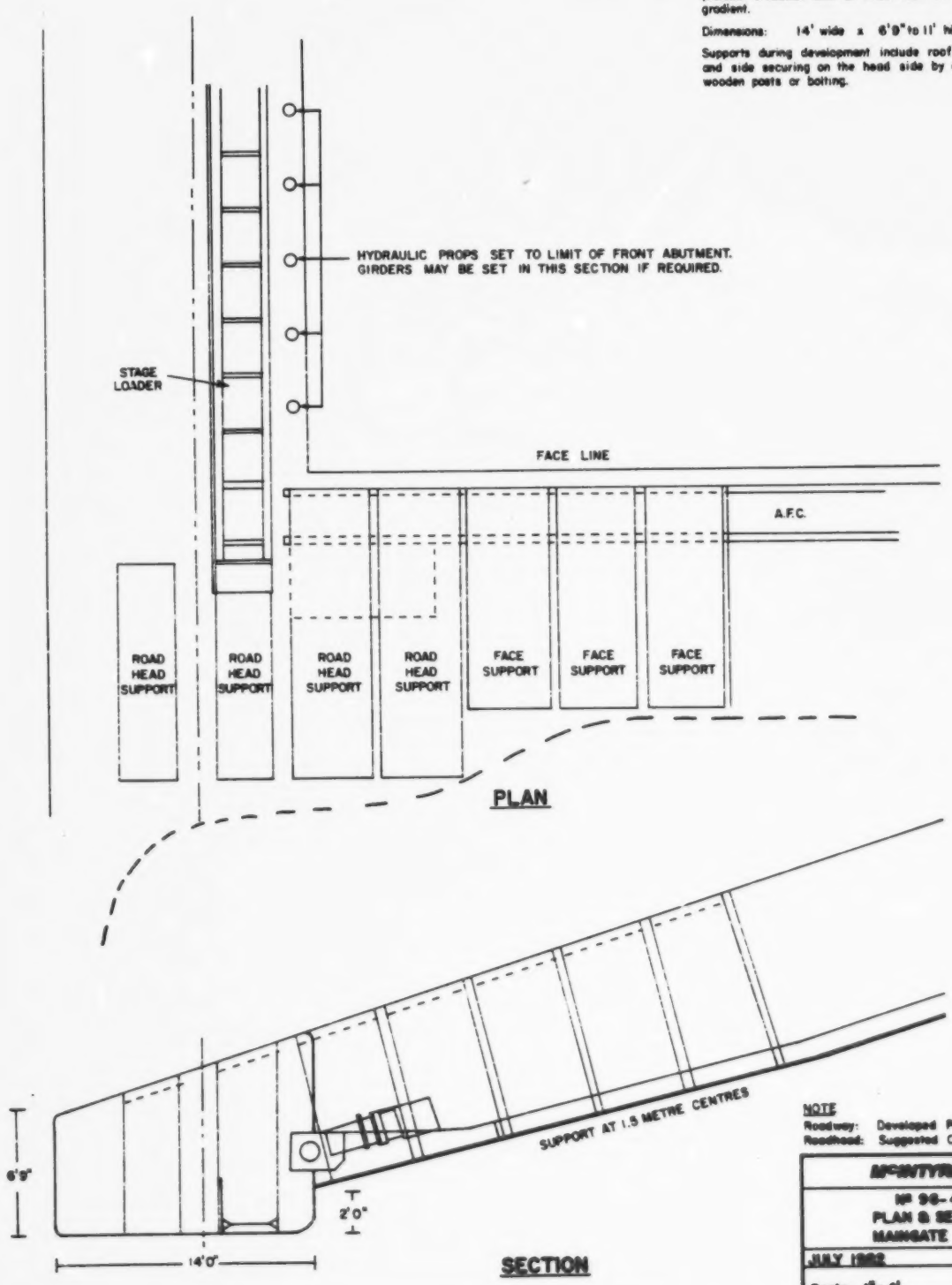

DAWSON CREEK
GRANDE PRAIRIE
PRINCE RUPERT
McINTYRE MINES
SASK.
PRINCE GEORGE
GRANDE CACHE
HINTON
EDMONTON
JASPER
ALBERTA
BRITISH COLUMBIA
CALGARY
VANCOUVER ISLAND
VANCOUVER
U . S . A .
LEGEND
ROADS
RAILWAYS
McINTYRE MINES
LOCATION MAP
JULY 1982 DRAWING Nº 1



NOTE: The maingate is developed using a profile miner to produce a section with an intact roof on seam gradient.

Dimensions: 14' wide x 6'9" to 11' high

Supports during development include roof bolting and side securing on the head side by either wooden posts or bolting.



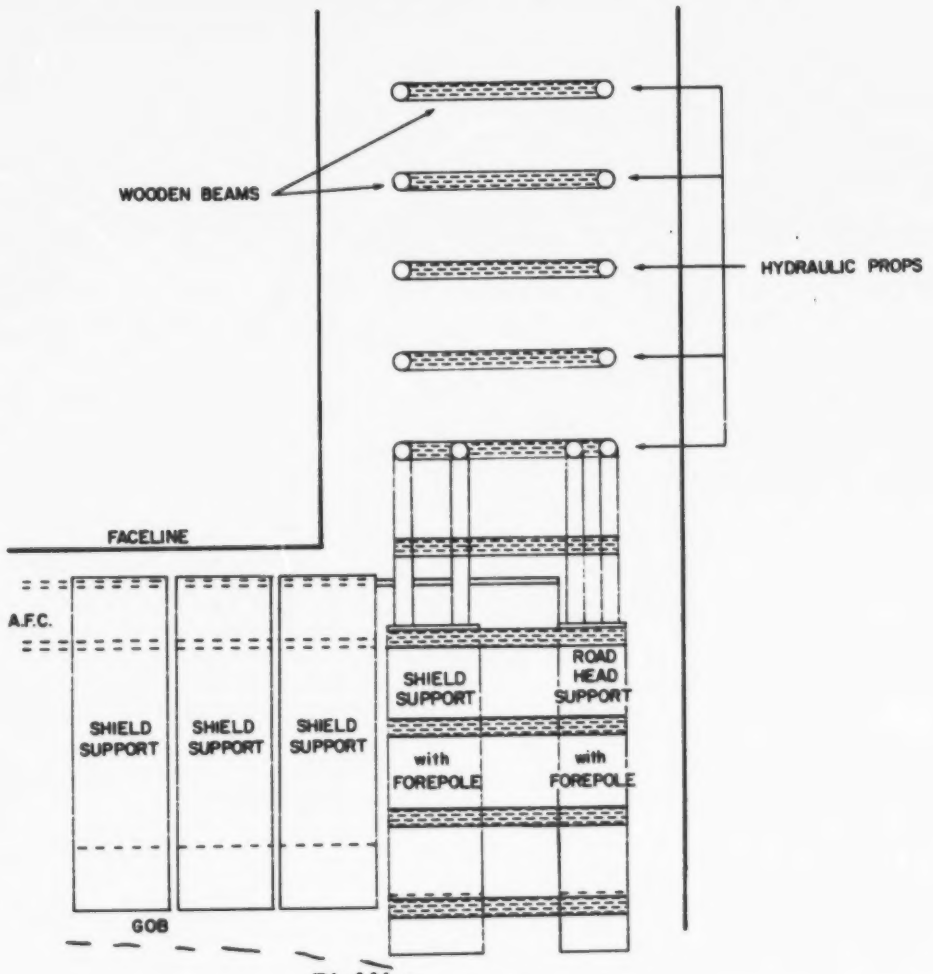
NOTE
Roadway: Developed Profile
Roadhead: Suggested Operational System

McINTYRE MINES

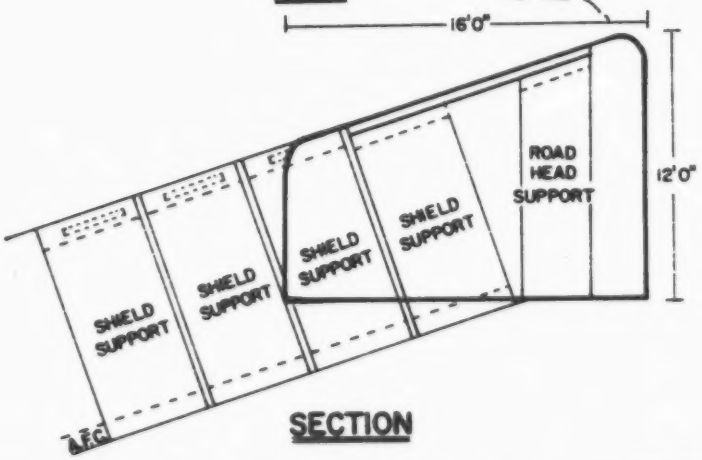
No 98-4 MINE
PLAN & SECTION OF
MAINGATE ROADHEAD

JULY 1982 DRAWING No 3

Scale: 1"=4'



PLAN



SECTION

NOTE
ROADWAY: Developed Profile
ROADHEAD: Suggested Operational System

McINTYRE MINES
№ 96 - 4 MINE PLAN & SECTION OF TAILGATE ROADHEAD
JULY 1982 DRAWING № 4
Scale: 1" = 4'

TABLE A

1. TECHNICAL DATA

POWERED SUPPORTS

1. Feed pressure (lb/in^2) .....
2. Open height (ft) .....
3. Closed height (ft) .....
4. Leg centres
   - (i) Face to waste (in) .....
   - (ii) Parallel to face (in) .....
5. Total weight (tons) .....
6. Support centres (in) .....
7. Support density for 27 in web .....
   - (i) Before cut ($tons/ft^2$) .....
   - (ii) After cut ($tons/ft^2$) .....
8. Distance between canopies (in) .....
9. Load per foot run of face ($tons/ft^2$) .....
10. Unsupported roof during advance (ft) .....
11. Average roof bearing at yield (lb/in^2) .....
12. Average floor bearing at yield (lb/in^2) .....
13. Base
    - (i) Thickness under legs (in) .....
    - (ii) Thickness in travel way (in) .....
    - (iii) Length (in) .....
    - (iv) Width (in) .....
    - (v) Total bearing area (ft^2) .....
14. Main canopy
    - (i) Thickness above legs (in) .....
    - (ii) Thickness in travel way (in) .....
    - (iii) Length (in) .....
    - (iv) Width (in) .....
    - (v) Bearing area of canopy (ft^2) .....
    - (vi) Maximum tip load at leg set (tons) .....
    - (vii) Maximum tip load at yield (tons) .....

15. Push out cantilever
    (i) Extend (in) .....
    (ii) Width (in) .....
    (iii) Maximum tip load (tons) .....
16. Anti-spall sprag
    (i) Extend (in) .....
    (ii) Plate area (in^2) .....
    (iii) Load (in^2) .....
17. Legs
    (i) Yield per leg (tons) .....
    (ii) Yield pressure (lb/in^2) .....
    (iii) Setting load at system pressure (lb/in^2) .....
    (iv) Cylinder diameter (in) .....
    (v) Cylinder bore (in) .....
18. Advancing ram
    (i) Stroke (in) .....
    (ii) Cylinder bore (in) .....
    (iii) Piston rod diam (in) .....
    (iv) Pull load at system pressure (lb/in^2) .....
    (v) Push load at system pressure (lb/in^2) .....
19. Support setting load per 1,000 (lb/in^2) operating pressure (tons) .....
20. System pressure (lb/in^2) .....
21. Support setting load at system operating pressure (tons) .....

2. TECHNICAL DATA

SHEARER

1. Weight (tons) .....
2. Height above floor (in) .....
3. Underframe height above conveyor (in) .....
4. Length (drum centres) (ft) .....
5. Maximum cutting height above floor level (ft) .....
6. Maximum cutting height below floor level (ft) .....
7. Total horsepower (hp) .....
8. Maximum pull (tons) .....
9. Corresponding haulage speed (ft per min) .....
10. Maximum haulage speed (ft per min) .....
11. Maximum pull at maximum haulage speed (tons) .....
12. Slow drum speed (rpm) .....
13. Fast drum speed (rpm) .....
14. Pick speed at slow (ft/sec) .....
15. Pick speed at fast (ft/sec) .....
16. Required power on drums on no load (hp) .....
17. Required power on hydraulics on no load (hp) .....
18. Maximum allowable torque on drum shaft (pd/ft) .....
19. Cooling water (gall/min) .....
20. (i) Water flow for pick jets (gall/min) .....
    (ii) Water flow for hsv .....
21. Water pressure for jets (lb/in^2) .....
22. Drums
    (i) Diameter (in) .....
    (ii) Width (in) .....
    (iii) Number of vanes .....
    (iv) Number of picks .....
    (v) Lacing .....
    (vi) Number of jets .....
    (vii) Barrel thickness (in) .....
    (viii) Vane thickness (in) .....
    (ix) Degrees of vane wrap (°) .....

3. TECHNICAL DATA

STAGE LOADER

1. Length of stage loader (ft) .....
2. Length of jib above belt conveyor (ft) .....
3. Pan length (in) .....
4. Pan width (in) .....
5. Pan height (in) .....
6. Deck plate thickness (in) .....
7. Number of pans .....
8. Chain dimension (mm) .....
9. Chain grade .....
10. Chain speed (ft/min) .....
11. Flight quantity .....
12. Flight spacing .....
13. Electric motor (hp) .....
14. Ratio of gearbox reduction .....
15. Maximum allowable torque on sprocket (ft lbs) .....
16. Number of teeth on sprocket (drive head) .....
17. Number of teeth on sprocket (face end) .....
18. Total conveyor weight (lb) .....
19. Pan weight (line and inspection) (lb) .....
20. Drive frame weight (lb) .....
21. Gearbox weight (lb) .....
22. Motor weight (lb) .....
23. Connector pan weight (lb) .....
24. Spill plate height and weight (in and lb) .....
25. Return end weight (lb) .....
26. Chain weight per length of assembly state (lb) .....
27. Estimated life of equipment .....

25. Weights (tons)
    (i) Motor unit .....
    (ii) Haulage unit .....
    (iii) Gearhead gearbox .....
    (iv) Outboard arm .....
    (v) Underframe .....
    (vi) Drum .....
    (vii) Cowl .....
26. Estimated life of shearer .....

4. TECHNICAL DATA

FACE CONVEYOR

1. Conveyor length (ft) .....
2. Pan width (in) .....
3. Pan length (in) .....
4. Pan height (in) .....
5. Deck plate thickness (in) .....
6. Number of pans (incl inspections) .....
7. Connector strength (tons) .....
8. Chain dimensions (mm) .....
9. Chain grade .....
10. Chain speed (ft/min) .....
11. Flight quantity .....
12. Flight spacing .....
13. Main gate motor power (hp) .....
14. Tail gate motor power (hp) .....
15. Ratio of gearbox reduction .....
16. Maximum allowable torque on sprocket (ft/lb) .....
17. Number of teeth on sprockets .....
18. Total weight of conveyors (tons) .....
19. Pan weight (line and inspection) .....
20. Drive frame weight (lb) .....
21. Gearbox weight (lb) .....
22. Motor weight (lb) .....
23. Connector pan weight (lb) .....
24. Ramp plate weight (lb) .....
25. Spill plate height from floor (in) .....
26. Goaf trapping weight if separate from spill plate (lb) .....
27. Chain weight per length of assembly (state) (lb) .....

5. TECHNICAL DATA

COAL CRUSHER

1. Maximum delviery (tons/hr) .....
2. Useful width (in) .....
3. Maximum lump dimensions (in x in x in) .....
4. Maximum final dimensions (in) .....
5. Electric motor power (hp) .....
6. Overall width (in) .....
7. Overall length (ft) .....
8. Overall height (ft) .....
9. Total weight (inc motor) (tons) .....
10. Speed of breaker element .....
11. Diameter of breaker element .....
12. Estimated life of crusher .....

6. TECHNICAL DATA

POWER PACK

1. Number of pumps .....
2. Pump capacity each (gall/min) .....
3. Working pressure (lb/in^2) .....
4. Maximum working pressure (lb/in^2) .....
5. Electric motor power (hp) .....
6. Number of tanks and dimensions .....
7. Tank Capacity (each) (gall) .....
8. Power pack unit size (ft x ft x ft) .....
9. Power pack unit weight (ton) .....
10. Total weight (ton) .....
11. High pressure hydraulic hoses
    (i) Inside diam (in) .....
    (ii) Bursting pressure (lb/in^2) .....
12. Return hydraulic hoses
    (i) Inside diam (in) .....
    (ii) Bursting pressure (lb/in^2) .....
13. Type of filtration .....
14. Protective devices .....

7. TECHNICAL DATA

HYDRAULIC PROPS

1. Quantity .....
2. Closed height without extension (ft) .....
3. Open height without extension (ft) .....
4. Closed height with extension (ft) .....
5. Open height with extension (ft) .....
6. Setting pressure (lb/in^2) .....
7. Setting load (tons) .....
8. Yield load (lb/in^2) .....
9. Yield pressure (lb/in^2) .....
10. Inside diameter of the outer cylinder (in) .....
11. Outside diameter of the outer cylinder (in) .....
12. Outside diameter of the inner member (in) .....
13. Prop weight (lb) .....
14. Extension weight (lb) .....

8. TECHNICAL DATA

ELECTRICAL EQUIPMENT

Remote Power Centre

Manufacturer

HV volts ..... at ..... h_3
HT amps .....
MT volts .....
MT amps .....
% voltage regulation at FL ..... at ..... pf
Physical dimensions ..... high x ..... wide x ..... long
Weight .....

HT Breaker

Breaking capacity .....
Type .....
Physical dimensions .....high x ..... wide x ..... long
(if not included above)
Weight ..... (if not included above)
Type of tripping circuit .....
Short circuit rating .....

Contactor Switchgear

Manufacturer

Nominal voltage ..... at ..... h_3

Incoming current capacity .....
Breaking cpacity of each contractor .....
Contactor type .....
Ground fault type .....
Ground check type .....
Over current type .....
Physical dimensions ..... high x ..... wide x ..... long
Weight ..... (State if per unit or overall)

Cables (for each type and size)

Type .....
Number of cores .....
Weight .....
Overall diameter .....
Type of plugs and receptacles .....

Signalling and Communication

Manufacturer

Console type .....
Nominal voltage .....
Dimensions .....
Weight .....
Number of keys .....
Number of amplifiers .....
Cable type(s) .....
Cable overall diameter .....

Lighting

Manufacturer

Type .....
Number of control units .....
Number of fittings .....
Voltages .....
Control unit dimensions .....
Control unit weight .....
Fitting dimensions .....
Fitting weight .....
Power .....
Output .....

TABLE B

DRAWINGS TO BE SUPPLIED

1. (i) Face layout showing equipment
   (ii) Cross section through the face:
      (a) After shear with conveyor advanced
      (b) After shear with supports advanced
      (c) After shear with supports advanced if cutting section has been reduced to height of 8 feet
   (iii) Section of intake return roadheads

2. Powered Supports

   (i) Arrangement drawing of support
   (ii) Arrangement of roadhead support
   (iii) Arrangement of tail gate anchor support
   (iv) Assembly drawing of legs and rams
   (v) Hydraulic layout
   (vi) Arrangement of hydraulic components
   (vii) Arrangement of stabilizing equipment
   (viii) Layouts showing movements of supports at various stages of operation

3. Shearer

   (i) Arrangement of the machine
   (ii) Drawings of the components
   (iii) Diagram of pick lacing on drum
   (iv) Drawings of picks
   (v) Haulage arrangement
   (vi) Pull force/speed diagram
   (vii) Layout of electrics
   (viii) Layout of hydraulics
   (ix) Auto control drawing
   (x) Drawing of cooling system
   (xi) Drawing of dust suppression system

4. Conveyors

(i) Drawing of drive heads and sprockets
(ii) Drawing of pans
(iii) Drawing showing chain assemblies
(iv) Chain tensioning device
(v) Drawing of pan connections
(vi) Drawings of ramp plate, spill plate/cable handler, M/C attachment and associated fittings
(vii) Drawing of stage loader - jib and return end assembly
(viii) Drawings of retraction system for the stage loader

5. Coal Crusher

(i) Arrangement drawing

6. Electrical Equipment

(i) General layout of full installation
(ii) Wiring and schematic diagrams of all items

7. Temporary Supports

(i) Arrangement drawing of prop

8. Hydraulic Power Pack

(i) Arrangement of pumps and tanks
(ii) Diagram of face hydraulic lines

TABLE C

PRICING DOCUMENT

ITEM	UNIT PRICE (Canadian)	QUANTITY	TOTAL PRICE (Canadian)
1. Longwall supports			
2. Buttress support (main gate)			
3. Buttress support (tail gate)			
4. Roadhead Supports			
5. Double ended shearer, complete with drums cable handler and lump breaker			
6. Longwall conveyor, with all accessories and motors			
7. Stage loader, with motor			
8. Coal crusher, with motor			
9. Communication and signalling system			
10. Lighting			
11. Hydraulic pumps, tanks, with motors and hydraulic hoses			
12. Electrical - switchgear cables, remote power centre			
13. Temporary props			
14. Spares			
15. Technical Assistance			
16. Training			
17. Compatibility Trials			

THE SUPPLY OF A COMPLETE LONGWALL MINING SYSTEM & SUPPORTING SERVICES

APPENDICES

AUGUST 1982

THE SUPPLY OF A COMPLETE LONGWALL MINING SYSTEM
& SUPPORTING SERVICES

APPENDICES

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APPENDIX 4 FLOOR STRENGTH

INTRODUCTION

McIntyre Mines has operated both surface and underground mines in the Smoky River for over fourteen years. During this time approximately 13.8 million long tons have been produced from underground mines mainly by room and pillar operations.

The mining areas have been extensively drilled and the geology of the area is well understood.

Although a significant number of geotechnical studies have been conducted they have usually been directed toward surface mining activity. Consequently little relevant background geotechnical data on underground mining operations exists. Recently, a program has been initiated to monitor caving characteristics and measure abutment pressures in the 9A-4 Mine during depillaring activity and a program to determine floor strengths at various horizons in the No. 4 seam has been completed. It is also envisaged that core samples of strata immediately above and below the coal seam and from within the seam will be obtained and subjected to testing and analysis by Seegmuller International.

SUMMARY

The geology of the area of the longwall faces is well proven and more information will be generated as the development continues.

The waste caves suitably as observed in No. 9A-4 Mine causing very limited problems due to forward abutment pressure.

The top 10 feet of the seam is relatively hard and the yield and ultimate strength of the floor coal down to 11.5 feet is in excess of 600 psi.

The depth of cover of 500 to 1000 feet may cause minor problems due to floor lift and side movement.

Roof water during development will need careful control to ensure minimum damage to the floor. Since the faces are to be retreated to the rise, the water will be left behind when in full production.

1. Use of Purchase Order Form

For Contracts involving the purchase of manufactured goods, ("off the shelf").

2. Purchase Order - Terms and Conditions Pertaining to the Procurement of Services

For Services of short duration and work of a very straight forward nature and for total consideration of less than $5,000.

Proposed Terms and Conditions for Purchase Order Forms

1. General

This order, including these terms and conditions, form the entire contract between the parties and supersede any prior agreements, whether written or verbal, and no variation thereof, irrespective of the wording of the Seller's acceptance, will be effective unless specifically agreed to in writing by the McIntyre. Any specifications, drawings, notes, instructions, engineering notices or technical data referred to in the purchase order shall be deemed to be included herein by reference as if fully set forth. In case of any discrepancies or inquires, Seller shall refer to McIntyre for decision, instructions, or for interpretation.

2. Quality

Seller warrants the quality and fitness for use of the goods, such goods to conform strictly to the specifications, drawings, or sample specified or furnished. All goods are subject to inspection by McIntyre within a reasonable time after actual delivery. McIntyre, at its option, may return to Seller goods rejected for failure to meet specifications or required quality of workmanship; or may perform such work as necessary to render them acceptable and Seller shall reimburse McIntyre for expenses of inspection and return shipment, or making goods acceptable and other costs in connection with rejected goods. No replacement shall be made without McIntyre's prior written instructions. Failure by McIntyre to make any inspection shall not relieve Seller of any of its obligations hereunder or any of the terms and conditions hereof. This warranty shall survive any inspection, delivery, acceptance or payment by McIntyre for the goods.

3. Acknowledgement and Acceptance

The Seller must acknowledge and accept the order on the form provided and return one signed copy within seven (7) days of receipt of the order. Failure to do so may invalidate the order, at the option of McIntyre. Commencement of performance pursuant to this purchase order constitutes acceptance hereof by Seller. If delivery dates cannot be met, Seller must inform McIntyre in

writing be return mail of Seller's best possible delivery for McIntyre's acceptance.

4. Deliveries

Time shall be of the essence of the contract. If deliveries are not made at the time agreed upon, McIntyre reserves the right to cancel or to purchase elsewhere, and hold Seller accountable therefor.

5. Risk

The goods shall be at the risk of the Seller who shall bear all loss, cost or damage from whatsoever cause arising, which may occur to the goods or any part thereof, until delivered to and accepted by McIntyre.

6. Packing and Cartage

No additional packaging, crating, or shipping charges will be allowed unless agreed to by McIntyre in writing and shown as a separate item on this order.

7. Patents

Seller shall indemnify McIntyre and hold it, its officers, agents and users of its product, harmless from and against any and all loss, liability or expense by reason of any claim or suit for alleged infringement of any copyright, trademark or patent, resulting from or arising in connection with the manufacture or sale, use or other disposition of any article or material furnished hereunder, or the performance of any work hereunder and shall defend any such claim or suit and pay all costs and expenses incidental thereto provided that McIntyre shall have the right, at its option, to participate in the defence of any such claim or suit without relieving Seller of any obligation hereunder.

8. Discounts

Discounts shall be calculated from the date when both the goods and acceptable invoices are received.

9. Laws

It is further agreed that the goods ordered above shall comply with all applicable Federal, Provincial and other laws and standards applying thereto. This purchase order and the acceptance thereof shall be a contract made in the Province of Alberta, and shall be governed by the laws thereof.

10. McIntyre's Property

All material, including tools, furnished or specifically paid for by McIntyre shall be the property of McIntyre, shall be subject to

removal at any time without additional cost upon demand by McIntyre, shall be used only in filling orders from McIntyre, shall be kept separate from other materials or tools and shall be clearly identified as the property of McINtyre. Seller assumes all liability for loss or damage, with the exception of normal wear and tear, and agrees to supply detailed statements of inventory at monthly intervals or as otherwise agreed upon.

11. Electrical Apparatus

All electrical apparatus must bear Canadian Standards Association approval or be on the C.S.A. approved list.

12. Foreign Shipments

Foreign shipments require certified customs invoices in quadruplicate.

13. Assignment

No right, title, estate or interest in this order and no obligation or duty or liability of the Seller shall be transferred, assigned, set-over, conveyed, pledged, hypothecated or otherwise disposed of without the prior written consent of McIntyre. Any attempts to so transfer, assign set over or convey, pledge, hypothecate, delegate, or any other method to avoid the provisions of this order shall be wholly void and totally ineffective for all purposes.

14. Taxes

Unless otherwise specifically provided, the Seller assumes exclusive liability for and shall pay before delinquent all sales, use, excise and other taxes, charges or other contributions of any kind, or with respect to or measured by the articles sold their material or work furnished hereunder for the wages, salary or other remuneration paid to persons employed in connection with the performance of this contract and Seller shall indemnify and hold McIntyre harmless from any liability and expense by reason of Seller's failure to pay such taxes or contributions. In cases where the Seller, due to Government regulations, can not exempt duty and/or sales tax from the invoice value, the rates and amounts of such sales tax and/or duty must be shown as a separate item on the invoice.

15. Set-off

McIntyre shall have the right, but not duty to withhold any monies payable to Seller for any indebtedness matured or unmatured arising out of this or any other of McIntyre's purchase orders with the Seller or its affiliates and apply same to the payment of any obligation of Seller to McIntyre or any other parties arising in any manner out of this contract or its performance.

16. Excuses For Non-Performance

Either party shall be absolved from its obligations under this contract when and to the extent that performance is delayed or prevented and in McIntyre's case, when and to the extent that its need for the articles, materials or work to be supplied hereunder is reduced or eliminated by any cause, except financial, reasonably beyond its control, including but not limited to fire, explosion, war, riots and strikes, labour disputes and government laws, orders or regulations.

17. Default

If Seller or any sub-contractor shall breach any provision hereof or shall become insolvent, enter voluntary or involuntary bankruptcy or receivership proceedings for the benefit of creditors, McIntyre shall have the right, without limiting any other rights or remedies which it may have hereunder, or by operation of law, to cancel any unfilfilled part of this purchase order without any liability whatsoever, and whereupon McIntyre shall be relieved of all further obligations hereunder. McIntyre's right to require strict performance of any obligation hereunder shall not be affected by any previous waiver, forbearance or course of dealing.

18. Notices

Any notice to be given by McIntyre under this order shall be deemed given one day after deposit in the mail, postage prepaid, addressed to Seller at its address shown on the face hereof.

19. Drawings and Specifications

McIntyre shall have the complete right to use for all its purposes, including the purchase or repair and replacement parts, all drawings and specifications provided to it hereunder by the Seller. McIntyre shall at all times have title to all drawings and specifications furnished by McIntyre to Seller and intended for use in connection with this purchase order. Seller shall use such drawings and specifications only in connection with this purchase order and shall not disclose such drawings and specifications to any person, firm or corporation other than McIntyre's or the Seller's employees, subcontractors or government inspectors. Seller shall, upon McIntyre's request or upon completion of this purchase order, promptly return all drawings and specifications to McIntyre.

20. Invoice Timing

Time consumed returning invoices for correction will constitute an extention of the discount period.

21. Guarantee

Seller, in addition to, but not in substitution for, any term or condition hereof, acknowledges and agrees that Mcintyre is relying on Seller's judgement and opinion that goods and any work upplied hereunder are capable of performing the function for which they are being supplied or done. No payment or acceptance by McIntyre shall constitute a waiver of the foregoing, nor shall herein contained be construed to exclude or limit any warranties construed or implied hereunder by law. All loss or damage to McIntyre directly and naturally resulting in the ordinary course of events from breach of any express of implied warranty with respect to the goods supplied hereunder shall be bourne by the Seller.

22. Indemnity

Subject to the preceding paragraphs, Seller shall hold McIntyre harmless from any and all loss, cost, damage, expense, claim, suit or liability on account of any and all damage to, or loss or destruction of any property including without limitation, property of Seller and McIntyre, or injury to or death of any person including without limitation employees of Seller or McIntyre arising directly or indirectly out of or in connection with the negligent performance of this contract. Seller shall defend any such claim or suit and pay all costs and expenses incidental thereto, but McIntyre shall have the right, at its option, to participate in the defence of any such claim or suit without relieving Seller of any obligation hereunder.

23. Prices

The Seller's price shall not be higher than last quoted or charged to McIntyre unless otherwise agreed in writing.

24. Quantities

Shipments must equal exact amounts ordered unless otherwise agreed by McIntyre.

WORK CONTRACT CONDITIONS

When this contract provides for the performance of any work, the following conditions in addition to the foregoing conditions, shall apply unless otherwise specified on the face hereof.

1. Performance

The Contractor shall diligently and carefully perform all work required hereunder in a good and workmanlike manner and shall furnish all labour, supervision, machinery, materials, equipment and supplies necessary therefor and if permitted to subcontract

shall be fully responsible for all work and services performed by subcontractors. The Contractor shall conduct all operations in the Contractor's own name as an independent contractor and not in the name of, or as agent for, McIntyre, and the Contractor's employees shall at all times be under its direction and control. Compliance by the Contractor, or its employees, with engineering directions, safety practices, maintenance instructions, or change orders issued by McIntyre or McIntyre's representative shall not affect the Contractor's status as an independent Contractor and shall not relieve the Contractor of the obligations assumed by it under this contract. No person who services or is supplied by Contractor hereunder shall be deemed an employee of McIntyre and the Contractor will indemnify McIntyre against all claims arising out of or in connection with the existence or termination of any such person's contract of employment with the Contractor.

2. Title and Responsibility

Title to all machinery, materials, equipment and supplies to be furnished to McIntyre by the Contractor hereunder will pass to McIntyre upon the arrival of the same at the worksite, but the Contractor shall be solely responsible for all loss, damage or destruction thereof and shall maintain in good operating condition the work in progress until the project is completed to McIntyre's satisfaction and replacement of items so lost, damaged or destroyed shall be accomplished promptly by the Contractor at its sole expense.

3. Insurance

The Contractor shall, throughout the term of this Contract, carry and cause its subcontractors to carry the following insurance in not less than the following amounts, with reputable and financially responsible insurance companies and shall furnish to McIntyre certificates specifying the names of the insurers, policy numbers and expiration dates establishing that such insurance is being procured and is being maintained.

(a) Workers' Compensation Insurance in compliance with the laws of the applicable jurisdiction;

(b) Employers' Liability Insurance covering each employee engaged in the operations herein to the extent of $500,000.00, where such employees are not covered by Workers' Compensation;

(c) Automobile Liability Insurance covering all motor vehicles owned or non-owned, operated and/or licences in the names of the Contractor with minimum bodily injury and property damage limits of $1,000,000.00 inclusive;

(d) Comprehensive General Liability Insurance including products and completed operations coverage with a minimum bodily injury and property damage limit of $1,000,000.00 inclusive.

4. Permits

(a) All necessary permits and licences shall be obtained by the Contractor, with the exception of permits and licences which are required in McIntyre's name, in which case the Contractor shall assist McIntyre in obtaining the same.

(b) The Contractor shall give all necessary notices, shall pay all fees and taxes required by law, and shall comply with all laws, ordinances, rules and regulations with respect to the work herein described.

(c) The Contractor shall comply with all applicable laws, ordinances, rules, and regulations with respect to labour employed on the work, including, without limitation, any applicable law pertaining to old age benefits, unemployment compensation, accident compensation, and health insurance; and shall pay any union dues or salary levels required in accordance with the rules and regulations of the United Steel Workers of America Union.

5. Use of Premises

The Contractor shall perform all work in such a manner as to cause minimum interference with McIntyre's operations and the operations of other contractors on the premises and shall take all necessary precautions (including those required by safety regulations) to protect the premises and all persons and property thereon from damage or injury and shall assume responsibility for the taking of such precautions by the Contractor's and subcontractor's employees, agents, licensees permittees and subcontractors. The Contractor shall employ, or cause to be employed, on or in connection with the performance of all work, only persons who are fit and skilled in the work assigned. The Contractor shall cause its employees and all subcontractors to abide by all safety and security rules in force on the premises, including compliance with the Occupational Health and Safety Act. Upon the completion of the work, the Contractor shall leave the premises clean and free of all tools, equipment, waste materials and rubbish.

6. Payments of Bills and Liens

The Contractor shall pay promptly all indebtedness for labour, materials, tools and equipment used in the performance of the contract before the Contractor shall be entitled to receive final payment. The Contractor shall furnish evidence satisfactory to McIntyre of the full payment of such indebtedness. The Contractor shall not permit any lien or charge to attach to the work or the premises upon which the work is being performed but if any shall so attach, the Contractor shall promptly procure its discharge and hold McIntyre harmless from all loss, cost, damage or expense incidental thereto. If at any time, McIntyre is informed of any such lien or charge, McIntyre shall have the right to retain out of any payment then due or thereafter to become due the Contractor under this contract, an amount sufficient to discharge such lien or charge.

7. Changes in Work

The Contractor shall make no change or alteration in the work, or perform additional work without McIntyre's specific written order.

8. Option to Terminate

McIntyre shall have the right to terminate this contract at any time regardless of whether the Contractor is in default by giving written notice to the Contractor, and in the event of any such termination, the Contractor shall receive as payment for the materials and work furnished and installed by the Contractor and accepted by McIntyre hereunder, such proportion as determined by McIntyre as the total contract price as the portion of the job which is completed and accepted bears to the entire job covered by this contract less any monies theretofore to the Contractor. The Contractor shall not in any event, be entitled to any prospective profits or damages because of such termination.

9. Cancellation

McIntyre reserves the right to cancel this contract at any time if:

(a) the performance is not undertaken and complete when specified or within a reasonable period of time if a completion date is not specified;

(b) Contractor repeatedly refuses or fails to supply enough properly skilled workmen, or equipment or materials of the proper quality or quantity to perform the work.

(c) Contractor fails in any respect to prosecute the work with sufficient promptness and diligence.

(d) The Contractor repeatedly fails to make timely payments to the Contractor's labour, subcontractors, materialmen or suppliers.

(e) The Contractor disregards laws, ordinances, government rules or regulations, or repeatly disregards instructions of McIntyre's representative which are consistent with the Contract.

10 Arbitration

Any controversy or claim arising out of or relating to this Contract or the breach thereof shall be determined by three arbitrators or a majority of them, one to be appointed by the Contractor and one by McIntyre, the third to be forthwith selected by the two so appointed. In the absence of agreement, the third shall be appointed pursuant to the Arbitration Act of Alberta. Decision of the Arbitrator shall be rendered within thirty (30) days after selection of the third arbitrator. Such decision is not rendered in such period, the provisions of this contract or arbitration shall be terminated, and any such controversy or claim shall be resolved by court action. Judgment upon any decision rendered in such arbitration proceeding may be entered in any court having jurisdiction thereof.

GEOLOGY

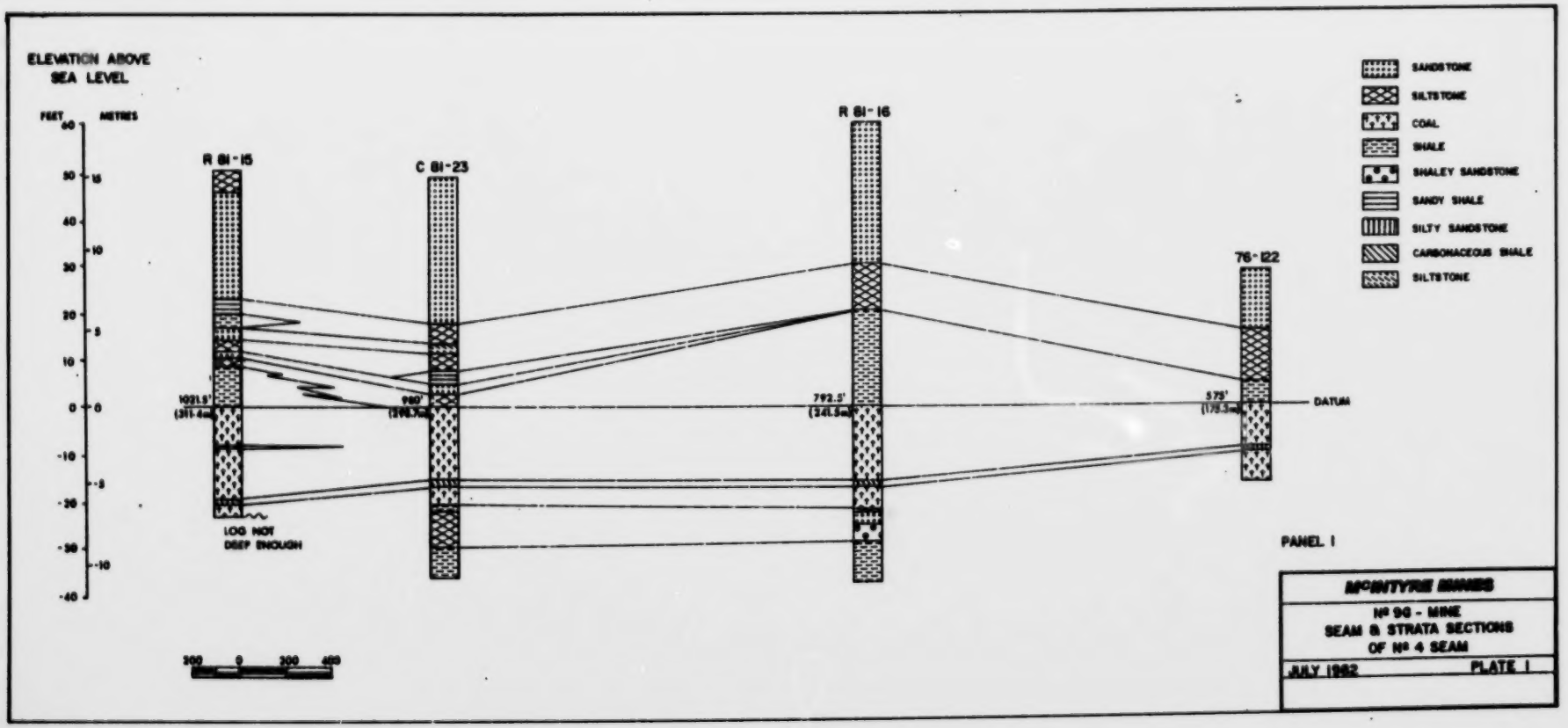
The coalfield lies in the Foothills of the Rocky Mountain thrust and fold belt. The area to be worked by No. 9G Mine has the Drill Camp anticline to the south and the Taylor syncline followed by a steep rise to the Barrett anticline to the north.

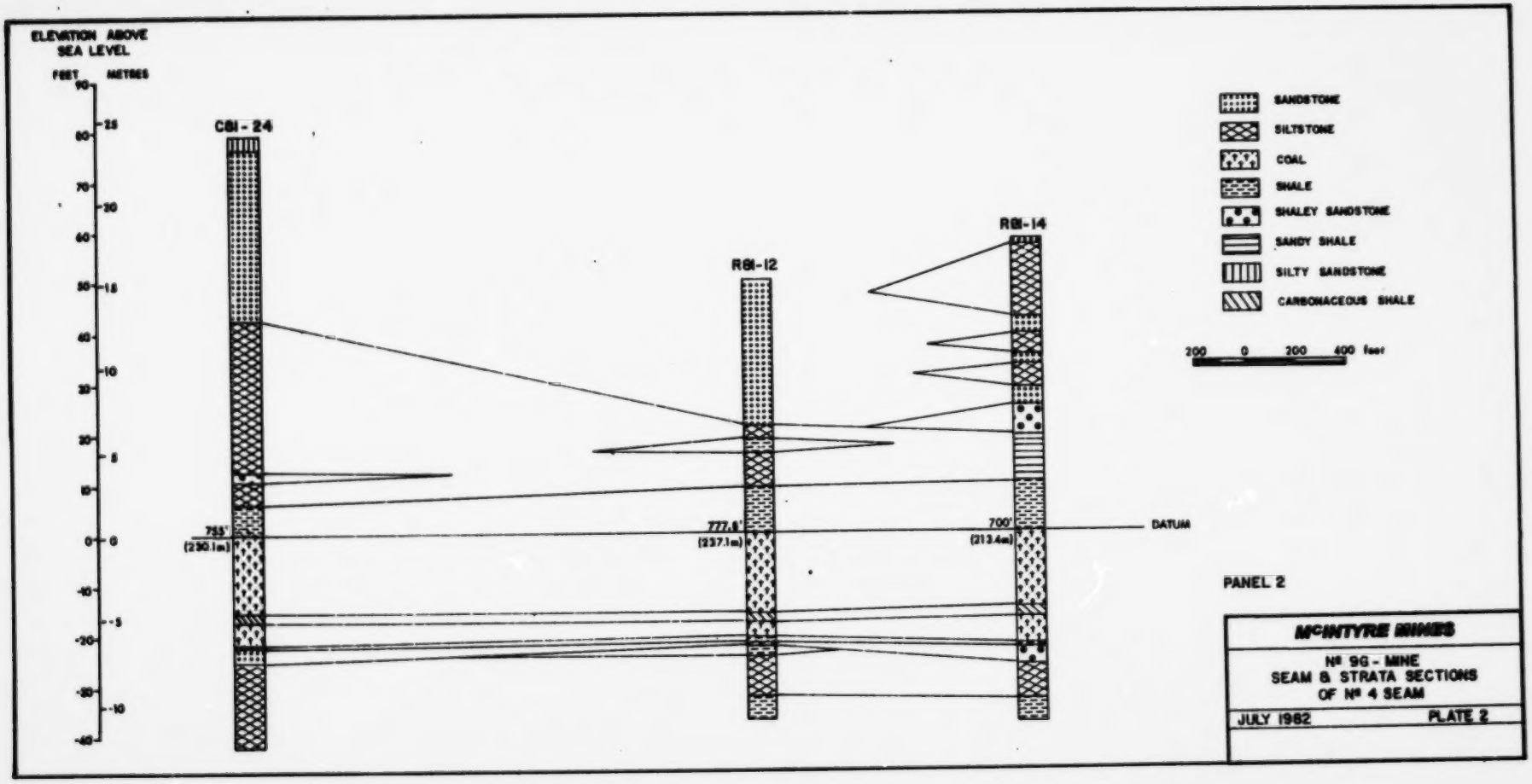
The coal seams were deposited in the Lower Cretaceous Period in a succession of interbedded alluvial sandstones, siltstones and mudstones. The strata has subsequently been deformed by folding and overthrusting in a northwesterly trend. Later general uplift and erosion due to glaciation took place to produce a rugged terrain at 4,000 feet with valleys trending northeasterly.

The longwall faces are to be developed in the area with the top of the Drill Camp anticline on the south and the bottom of the Taylor syncline to the north. (Drawing No. 1). The general dip of the strata in this area is from 20° to as little as 4° and then to 18° as the syncline is approached. General strata dip is from south to north with strike on an east west orientation.

The main coal seams recently worked underground are Nos. 4 and 11 with a minor development into No. 10 seam at No. 9G Mine. The proposed longwall faces will be in No. 4 seam of No. 9G Mine and the sections showing lithological variation in strata above and below the seam (Plates 1 and 2) are based on geophysical logs and core descriptions of drill holes. (Drawing No. 1).

Gradients along the face will vary from 20° to 4° on the first face but will be fairly uniform at about 18° on the second face.


ELEVATION ABOVE SEA LEVEL
FEET
METRES
60
50
40
30
20
10
0
-10
-20
-30
-40
15
10
5
0
-5
-10
R 81-15
C 81-23
R 81-16
76-122
1021.5'
(311.4m)
980'
792.5'
(241.5m)
575'
DATUM
LOG NOT DEEP ENOUGH
200
0
200
400
SANDSTONE
SILTSTONE
COAL
SHALE
SHALEY SANDSTONE
SANDY SHALE
SILTY SANDSTONE
CARBONACEOUS SHALE
SILTSTONE
PANEL I
McINTYRE MINES
No 96 - MINE
SEAM & STRATA SECTIONS
OF No 4 SEAM
JULY 1982
PLATE I


ELEVATION ABOVE SEA LEVEL
FEET
METRES
CBI - 24
RBI-12
RBI-14
755'
(230.1m)
777.8'
(237.1m)
700'
(213.4m)
DATUM
SANDSTONE
SILTSTONE
COAL
SHALE
SHALEY SANDSTONE
SANDY SHALE
SILTY SANDSTONE
CARBONACEOUS SHALE
200 0 200 400 feet
PANEL 2
McINTYRE MINES
Nº 9G - MINE
SEAM & STRATA SECTIONS
OF Nº 4 SEAM
JULY 1982
PLATE 2

Based on an analysis of data derived from exploration programs, faulting is not expected. Additional information will be derived as the faces are developed, however, small folds or local rolls may occur throughout the area.

The floor of the No. 4 seam is hard, though it has rarely been exposed by working or drilling. The strata is a succession of siltstones and sandstones with compressive yield strengths over 1000 psi.

The seam itself varies from 20 feet to 26 feet in thickness. The upper section to 12 feet from the roof is clean, good quality and quite competent. The coal below this section contains a band with a higher ash content and a sheared coal section of between 1 foot and 2 feet 6 inches.

The sheared coal (locally referred to as "Black Jack") is very soft and of very low compressive strength as a result of tectonic disturbances following deposition.

In general, the lower section of the seam is not as strong as the upper section.

The immediate roof is of mixed coal and shale, (caprock), from 0 to 2 feet thick, which often falls during working. The overlying strata is a mixture of mudstones, siltstones and interbedded sandstones with local shale partings.

There is no particular cleat in the coal and fracturing varies with the position of the coal in the structures.

GEOTECHNICAL

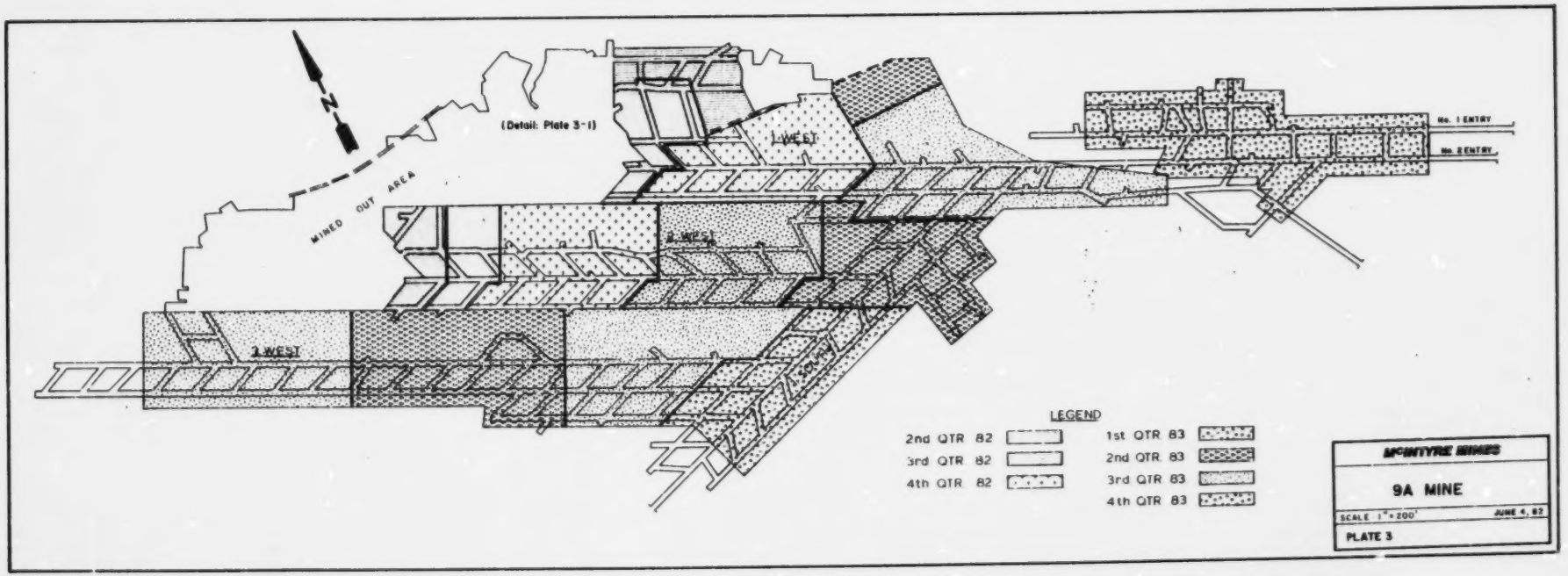
There is limited geotechnical and strata control information available from previous underground mining activity.

Recently underground strata control observations have been carried out in No. 9A-4 Mine which is working coal adjacent to No. 9G-4 mine. Three sections of coal have been developed by room and pillar methods over a period of 3 years. Depillaring has now commenced and an area 500 feet wide by over 3,000 feet long will be extracted. Tests are being conducted to measure the forward abutment pressure by the use of strategically placed load cells. At the same time examination of waste caving and pillar deformation are recorded.

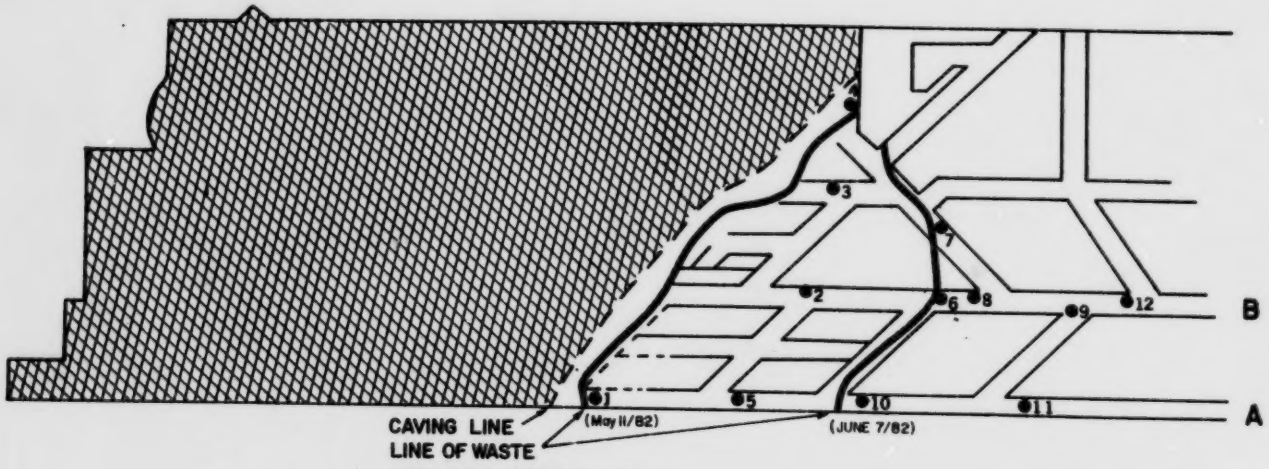
The load cells show conditions similar to those expected in longwall working. The load adjacent to the pillar side does not exceed 400 pounds per square inch while the load on the centre line of the area reaches 1600 pounds per square inch.

In general, the waste is caving in two phases. Initially, up to about 10 feet above the coal, which does not fill the waste void. Later, a second cave occurs which may fill the void completely. However, observation is not possible from a position of safety. (Plate 3 and 3-1)

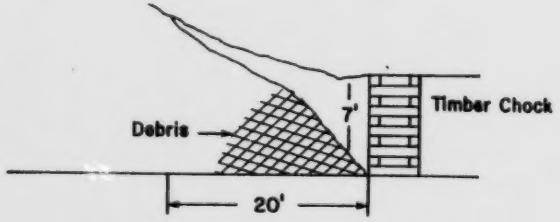
Roadway closure has been very limited despite the action of roof water during development. The pillars have suffered slight deformation and generally the area is caving satisfactorily with no heavy front abutment pressures being produced.

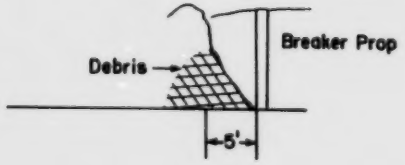

N
(Detail: Plate 3-1)
MINED OUT AREA
1 WEST
2 WEST
3 WEST
SOUTH
No. 1 ENTRY
No. 2 ENTRY
LEGEND
2nd QTR 82
3rd QTR 82
4th QTR 82
1st QTR 83
2nd QTR 83
3rd QTR 83
4th QTR 83
McINTYRE MINES
9A MINE
SCALE 1"=200'
JUNE 4, 83
PLATE 3

DEPILLAR AREA SHOWING LINE OF WASTE AND SECTIONS ALONG ROADWAYS.

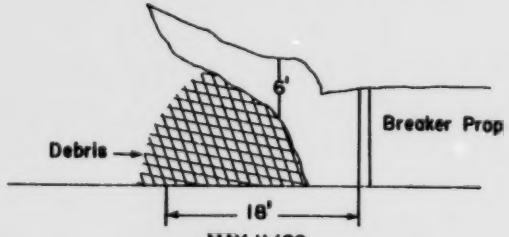


SECTION A

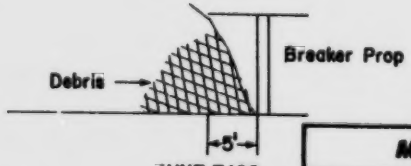




SECTION B



MAY 11/82



JUNE 7/82

McINTYRE MINES
NO. 9A - 4 MINE I-WEST DISTRICT
AUGUST 1982 PLATE 3-1

An evaluation of the base mechanical properties of the rock types was carried out by R.M. Hardy & Associates Ltd. and a copy of that information is attached.

Cover

The strata cover at No. 9G-4 Mine will vary from 500 to 1000 feet on the initial faces and pillar sizes have been designed to withstand the expected pressures.

The development roadways will be driven 16 feet in width as distinct from the current room and pillar roads which are 20 feet wide.

Hydrology

The ingress of water is anticipated during the development of the longwall faces. During the development of mines 9A and 9G water has been released from the area adjacent to the back of the development headings.

The source of the water has not been identified as being either aquifers in the super adjacent strata, percolating surface water or a combination of both. The quantity of water being produced in No. 9G Mine development has been estimated to be in excess of 50 gallons per minute.

Support of the Longwall Face

Problems attributed to poor support performance along the coal face and failure to adequately cave the waste are quoted as the main cause of failure of the two previous attempts to work the longwall system at McIntyre's Smoky River operation.

The supports must be capable of supplying sufficient resistance to cause the roof strata to break and that resistance must be built up through the floor on which the supports sit.

Tests of the yield and ultimate strengths of the coal have been carried out by the Coal Mining Research Council, CMRC. The results show that the top 11.5 feet of strata have yield and ultimate strengths of +600 psi and +900 psi respectively.

If only the top 10 feet of coal is taken, the coal floor should be competent for suitably designed supports.

REPORT ON
GEOTECHNICAL AND HYDROLOGICAL FACTORS RELATIVE TO PROPOSED NO. 9 MINE EXTENSION
MCINTYRE MINES LTD.
GRANDE CACHE ALBERTA

PREPARED FOR

MCINTYRE MINES LTD.

GRANDE CACHE, ALBERTA

BY

R. M. HARDY & ASSOCIATES LTD.

CALGARY ALBERTA

JUNE 18, 1976

3-6

LABORATORY TESTING

Laboratory testing was undertaken to provide basic mechanical rock properties of significant rock types for analytical design and stability evaluation of proposed mine pit walls in the No. 9 Mine Extension, Grande Cache, Alberta.

Core (WX size) was obtained in the field from drill hole 75-94 in the intervals 284 to 312 ft (below No. 10 seam) and 584 to 613 ft (below No. 4 seam). It was classified visually according to lithological type into medium grained sandstone, fine grained sandstone, siltstone, and mudstone. Representative samples of these rock types were selected for testing. Tests, conducted in R. M. Hardy & Associates Ltd. laboratory in Calgary, included bulk density, bulk specific gravity, porosity, slake-durability, point load strength, compressive strength, Young's modulus, Poisson's ratio and direct shear strength. Each test was run in accordance with either the American Society for Testing and Materials (A.S.T.M.) specifications or the International Society for Rock Mechanics (I.S.R.M.) "Suggested Methods". These are listed in Table I.

TABLE I

Rock Testing Specifications

Test	Specification Used	
Bulk Density	I.S.R.M. [1]	1972
Bulk Specific Gravity	I.S.R.M. [1]	1972
Porosity	I.S.R.M. [1]	1972
Slake-Durability	I.S.R.M. [2]	1972
Point Load Strength	I.S.R.M. [3]	1972
Compressive Strength	I.S.R.M. [3]	1972
Young's Modulus	A.S.T.M. [4]	C469-65
Poisson's Ratio	A.S.T.M. [4]	C469-65
Direct Shear Strength	I.S.R.M. [5]	1974

1. Part 1. Suggested Methods For Determining Water Content, Porosity, Density, Absorption, and Related Properties.

2. Part 2. Suggested Methods For Determining Swelling and Slake-Durability Index Properties.

3. Suggested Methods For Determining the Uniaxial Compressive Strength of Rock Materials and the Point Load Strength Index.

4. Static Modulus of Elasticity and Poisson's Ratio of Concrete in Compression.

5. Suggested Methods For Determining Shear Strength.

Additional relevant test results from a previous study done by R. M. Hardy & Associates Ltd. for McIntyre Porcupine Mines Ltd. in 1973 have been included here due to the similar strata encountered and the close proximity of the two areas under investigation.

Bulk density test results are summarized and presented in Table 2. Bulk density is the ratio of the in-situ weight to bulk volume. Test results suggest that fine and medium grained sandstones are generally more dense than siltstones or mudstones.

Slake-durability, porosity and bulk specific gravity test results appear in Table 3. The slake-durability test assesses the effect of weathering under simulated wetting and drying processes. It is expressed as a ratio of the amount of sample retained after several wetting and drying cycles to the initial sample weight. The results show that slake-durability varies from very high (sandstones) to medium high (siltstones) to medium low durability (mudstones).

Porosity is the ratio of pore volume to bulk volume of a sample. Rocks with high porosity tend to be weak due to the presence of voids within the rock mass.

TABLE 2

GRANDE CACHE

BULK DENSITY TEST RESULTS

Rock Type	Test Hole	Depth Feet	Depth (m)	Bulk Density pcf	Bulk Density t/m^3	Average Bulk Density pcf	Average Bulk Density t/m^3
Sandstone	75-94	592	(180.4)	168.2	(2.69)		
	75-94	600	(182.9)	165.8	(2.66)		
	75-94	602	(183.5)	168.0	(2.69)		
	75-94	307	(93.6)	166.1	(2.66)		
	75-94	298	(90.8)	169.4	(2.71)		
	75-94	301	(91.7)	163.0	(2.61)		
	104	51	(15.5)	168.7	(2.70)	167.7	(2.69)
	104	96	(29.3)	166.8	(2.67)		
	73-45	94.5	(28.8)	168.1	(2.69)		
	73-45	159	(48.5)	169.0	(2.71)		
	14	15.5	(4.7)	170.0	(2.72)		
	14	17	(5.2)	169.3	(2.71)		
Siltstone	75-94	285	(86.9)	156.9	(2.51)		
	75-94	288	(87.8)	161.7	(2.59)	160.4	(2.57)
	75-94	587	(178.9)	162.7	(2.61)		
Mudstone	75-94	308	(93.9)	150.2	(2.41)		
	75-94	310	(94.5)	162.0	(2.59)	158.0	(2.53)
	75-94	605-609	(184.4-185.6)	161.9	(2.59)		
Coal (weak, wet)	12	15.5	(4.7)	83.6	(1.34)	83.7	(1.34)
	12	16.5	(5.0)	83.8	(1.34)		
Coal, (strong, banded)	70-23	not known	not known	93.5	(1.50)	93.5	(1.50)

TABLE 3

Slake-Durability, Porosity, and Bulk Specific Gravity Test Results

Rock Type	Depth ft (m)	Slake-Durability %	Porosity %	Bulk Specific Gravity
Sandstone (medium)	592 (180.4)	99.4	0.29	2.69
	600 (182.9)	98.9	0.62	2.66
Sandstone (fine)	307 (93.6)	98.7	1.04	2.66
	298 (90.8)	99.2	1.82	2.72
Siltstone	285 (86.9)	91.4	5.40	2.51
	587 (178.9)	92.5	1.72	2.61
Mudstone	310 (94.5)	88.0	3.91	2.60
	605-609 (184.4-185.6)	18.0[1]	2.74	2.59

[1] Very low durability caused by extensive hairline fractures throughout the sample.

NOTE: Samples obtained from test hole 75-94.

The results indicate that all rock types have low to very low porosities. Bulk specific gravity is the ratio of the weight of a sample of given volume to the weight of an equal volume of water. The results of Table 3 are consistent with previously obtained values on similar sedimentary rocks.

A summary of the Point Load Strength Index results are presented in Table 4 and the complete test results are shown in Table 5. Point load results are used to supplement those of the unconfined uniaxial compression test. Advantages of the point load test include speed, economy and that virtually no sample preparation is required. The test involves breaking either rock core or irregular lumps between two conical loading platens. As strong layering is inherent in sedimentary rock strata, the load was applied parallel and perpendicular to bedding to develop an index of anisotropy. The results show a point load strength range of extremely high to very high for loading perpendicular to bedding and very high to high for loading parallel to bedding. The samples are highly anisotropic with point load strength perpendicular to bedding approximately four times greater than strength parallel to bedding.

Uniaxial compressive strength test results are shown in Table 6. The compressive strength is the loading force required to fail the sample divided by the average cross-sectional area of the sample. The results indicate that sandstone and siltstone are strong and that

TABLE 4

AVERAGE POINT LOAD STRENGTH INDEX RESULTS

Rock Type	$I_s(50)$ Parallel to bedding psi (MPa)	$I_s(50)$ Perpendicular to bedding psi (MPa)	$I_a(50)$ Anisotropy Index
Sandstone (medium)	667 (4.6) [1]	2436 (16.8) [3]	3.6
Sandstone (fine)	522 (3.6) [5]	1551 (10.7) [4]	3.0
Siltstone	217.5 (1.5) [9]	1348 (9.3) [10]	6.2
Mudstone	217.5 (1.5) [11]	449 (3.1) [10]	2.1

[ ] indicates number of samples tested

Note: samples obtained from test hole 75-94

POINT LOAD STRENGTH INDEX SCALE[1]

$I_s(50)$ MPa	Strength
0.01-0.03	extremely low
0.03-0.1	very low
0.1 -0.3	low
0.3 -1	medium
1 -3	high
3 -10	very high
10 -30	extremely high

1 Franklin, J.A. , "Classification of Rock According to its Mechanical Properties", Ph.D., Thesis, University of London, 1970.

TABLE 5

R.M. HARDY & ASSOCIATES LTD.

POINT LOAD STRENGTH INDEX RESULTS

Parallel to Bedding

Depth Feet	$I_s(50)$ psi	$I_s(50)$ (MPa)
Sandstone (medium)		
601	667	4.6
Sandstone (fine)		
598	167	1.15[1]
298	536	3.7
602-605	884	6.1
602-605	493	3.4
596	536	3.7
Siltstone		
288	0	0.0
285	0	0.0
599	0	0.0
602-605	319	2.2
587	522	3.6
298	333	2.3
296	246	1.7
290	362	2.5
588	217	1.5
Mudstone		
605-609	0	0.0
605-609	0	0.0
605-609	0	0.0
609-613	0	0.0
609-613	1696	11.7[2]
310	0	0.0
605-609	0	0.0
585	580	4.0
292	33	0.23
605-609	103	0.71
305	0	0.0

Perpendicular to Bedding

Depth Feet	$I_s(50)$ psi	$I_s(50)$ (MPa)
Sandstone (medium)		
592	15.0	15.0
602	17.0	17.0
601	18.5	18.5
Sandstone (fine)		
602-605	1276	8.8
602-605	1392	9.6
602-605	1870	12.9
596	1638	11.3
Siltstone		
290	884	6.1
290	1015	7.0
599	1914	13.2
587	1015	7.0
288	1421	9.8
288	884	6.1
298	1015	7.0
296	826	5.7
290	2900	20.0
588	165	11.4
Mudstone		
609-613	138	0.95
609-613	52	0.36
605-609	710	4.9
605-609	942	6.5
609-613	275	1.9
605-609	0	0.0
605-609	0	0.0
585	1305	9.0
292	449	3.1
605-609	638	4.4

1 Failure along coaly parting

2 Extremely well-cemented with calcite

Note: Samples obtained from test hole 75-94

TABLE 6

UNIAXIAL COMPRESSIVE STRENGTH

TEST RESULTS

Rock Type	Test Hole	Depth Feet	Depth (m)	Height Width Ratio	Compressive Strength psi	Compressive Strength (MPa)	Average psi	Average (MPa)
Sandstone (fresh)	75-94	598	(182.3)	1.5	8260	(57.0)		
	104	51	(15.5)	1.0	16914	(116.6)		
	104	96	(29.3)	1.0	11336	(78.2)		
	73-45	945	(28.8)	1.0	11762	(81.1)	11387	(78.5)
	73-45	159	(48.5)	1.0	16305[1]	(112.4)		
	75-94	592	(180.4)	2.0	12730[1]	(87.8)		
	14	15.5	(4.7)	1.0	10333[1]	(71.3)		
	14	17	(5.2)	1.0	3459[1]	(23.8)		
Siltstone (fresh)	101	91	(27.7)	1.0	15631	(107.8)		
	103	104	(31.7)	1.0	8365[1]	(57.7)	8865	(61.1)
	103	41	(12.5)	1.0	2598[1]	(17.9)		
Mudstone (fresh)	105	103	(31.4)	1.0	1141	(7.9)	1141	(7.9)
Coal (weak,wet)	12	15.5	(4.7)	1.0	5.5	(0.0)	5.5	(0.0)
	12	16.5	(5.0)					
Coal (banded, strong)	70-23	not known		1.0	1137	(7.8)	1137	(7.8)

1 Sample is slightly weathered and altered.

mudstone and coal are moderately weak according to Deer and Miller's classification[1].

A linear relationship was assumed between uniaxial compressiye strengths and point load strength indices as illustrated on Plate 1. Linear Regression Analysis gave slopes of 7.1 and 50.0 for loading perpendicular and parallel to bedding respectively. Broch & Franklin[2] determined a relationship $\sigma_c \cong 24 \times I_s(50)$, however, this was for isotropic sandstone.

Results of static elastic testing are summarized in Table 7. The modulus of elasticity was calculated by averaging the deformation of two diametrically opposite gages each parallel to the axis and each centered about mid height on the specimen. The averaged deformations at selected loads were plotted and the slope of the resulting line determined. Poisson's Ratio is the tangential (circumferential) strain divided by the axial (longitudinal) strain. The load versus longitudinal strain and load versus tangential strain curves are presented on Plates 2 to 9.

[1]Deere, D.U. & Miller, R.P. "Engineering Classification and Index Properties for Intact Rock". Tech. Rept. No. AFWL-TR-65-116., Air Force Weapons Lab., Kirtlavel Air Force Base, New Mexico, 1966.

[2]Broch, E. and Franklin, J.A., "The Point Load Strength Test", Trans. Inst. Min. Metall., V9, pg. 669, 1972.

TABLE 7

Static Elastic Properties

Sample No.	Rock Type	Modulus of Elasticity E_s psi (MPa)	Poisson's Ratio N_s
592	Sandstone (medium)	8.1×10^6 (55.9×10^3)	0.14
298	Sandstone (fine)	7.4×10^6 (51.0×10^3)	0.12
598	Sandstone (fine)	7.8×10^6 (53.8×10^3)	0.16
288	Siltstone	4.8×10^6 (33.1×10^3)	0.16
298[1]	Sandstone (fine)	6.6×10^6 (45.5×10^3) C_p = 150 psi	
		7.5×10^6 (51.7×10^3) C_p = 750 psi	

[1] Triaxial test results at confining pressures (C_p) of 150 psi and 750 psi.

NOTE: Samples obtained from test hole 75-94.

Elastic properties under triaxial loading were determined on specially prepared sandstone core. Two cycles were run at confining pressures of 150 psi (1.0 MPa) and 750 psi (5.2 MPa). Results for the elastic modulus

shown in Table 7 show a tendency for the modulus to increase with increasing confining pressure. Load versus strain curves are contained on Plates 10 and 11.

Peak and residual shear strength characteristics of the main rock types were determined by means of direct shear tests using a direct shear machine with an 8 inch square box. The specimens were prepared rock cores that were cast into plexiglass moulds which in turn were fitted into the shear apparatus. Each sample was run at three or four different normal stresses to develop the required shear strength parameters. The test results are summarized in Table 8 together with previously reported data from the same general area. The shear stress versus horizontal displacement, normal stress versus shear stress, and vertical displacement versus horizontal displacement curves for each rock type are presented on Plates 12-15.

The shear surfaces in the present testing were in all cases approximately parallel to bedding and are described in Table 8. The effective roughness angle or angle of surface asperities was also estimated.

TABLE 8

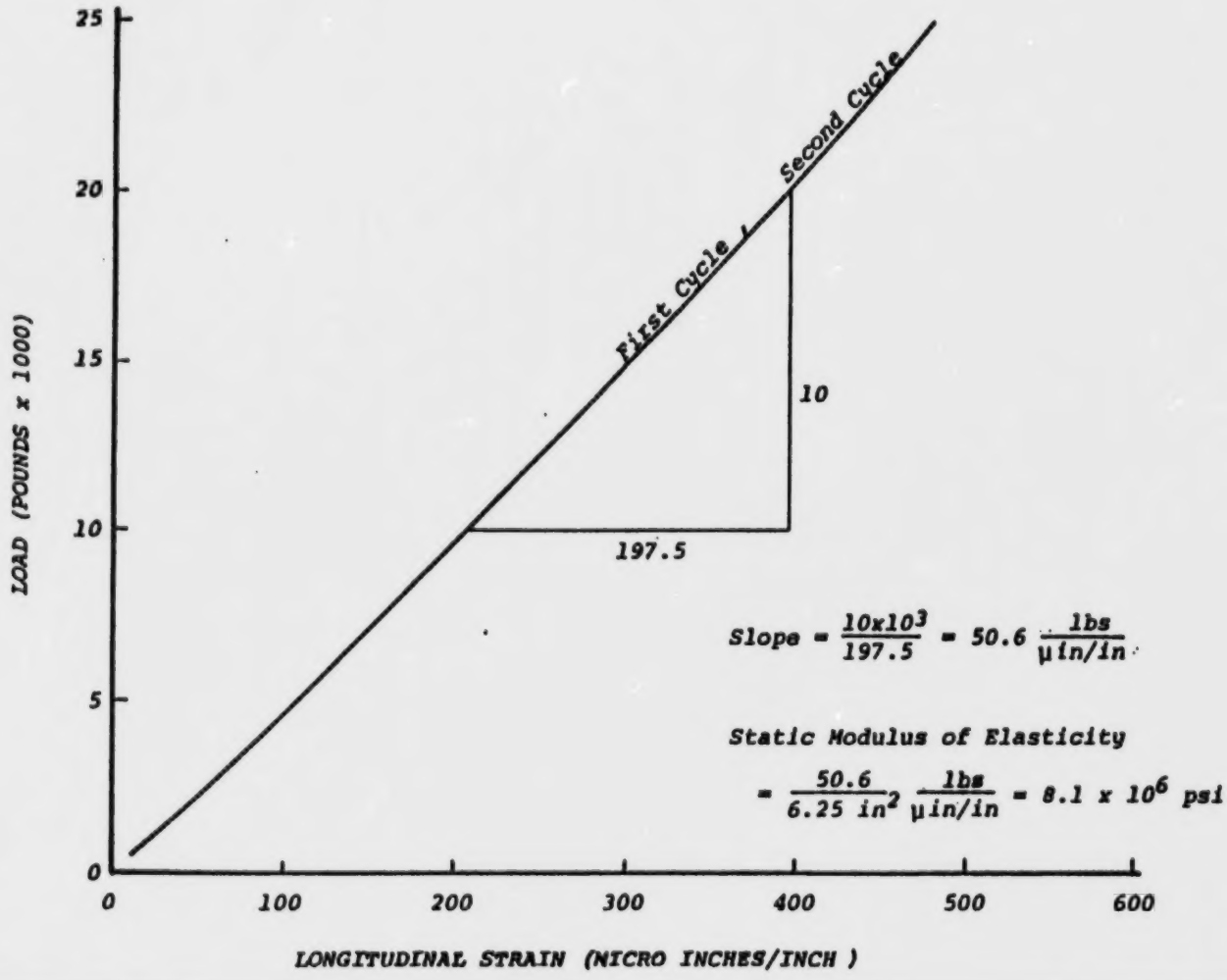
DIRECT SHEAR TEST RESULTS

Sample Description	Characteristics of Shear Plane	Residual Parameters		Peak Parameters		Effective Roughness Angle
		ϕ_r	C_r psi.	ϕ_p	C_p psi.	i
Sandstone, medium grained, a few carbonaceous coatings	broken approximately parallel to bedding rough, irregular surface	46°	0	47°	14	10 - 15°
Carbonaceous mudstone containing coaly partings	shiney coaly parting broken parallel to bedding, surface is undulating and rough	29°	1	28°	16.5	2 - 4°
Siltstone, sandy, finely bedded	broken parallel to bedding, surface is undulating, rough	37°	8.5	42°	33	5 - 7°
Sandstone, fine-grained, moderately weathered	natural joint parallel to bedding, weathered surface is rough and undulating	33°	2.5	-	-	5°
Sandstone,[1] fresh, fine grained, strong	parallel to bedding, rough surface	37°	9.0	41°	15.0	9°
Sandstone, fresh[1], medium grained, strong	joint perpendicular to bedding surface	28°	1.5	23°	15.3	9°
Siltstone, fresh[1], moderate strength	parallel to bedding, even surface	31°	7.0	38°	13.6	6°
Mudstone, fresh[1], moderately strong carbonaceous	bedding plane, smooth surface	20°	2.5	-	-	6 - 10°
coal[1], parting in mudstone	bedding plane within a coal seam, smooth surface	15°	3.0	16°	4.7	6°

1 Shear stress versus displacement and shear stress versus normal stress curves are contained in 1973 report.

R.M.HARDY & ASSOCIATES LTD.

Shear test results are governed by the characteristics of the shear plane, the rock type, and the rate of shearing. Generally, all surfaces were fresh and undisturbed resulting in relatively high peak strengths, however, sample S-1 exhibited vitually no peak on the displacement vs shear stress curve suggesting previous movement along the surface in the geological past. Weathering may also have produced the same effect. Evidence suggests that laboratory shear results are lower than actual insitu values since laboratory testing does not include the effects of large scale discontinuities present on failure surfaces in the field.



$$\text{Slope} = \frac{10 \times 10^3}{197.5} = 50.6 \ \frac{\text{lbs}}{\mu\text{in/in}}$$

Static Modulus of Elasticity

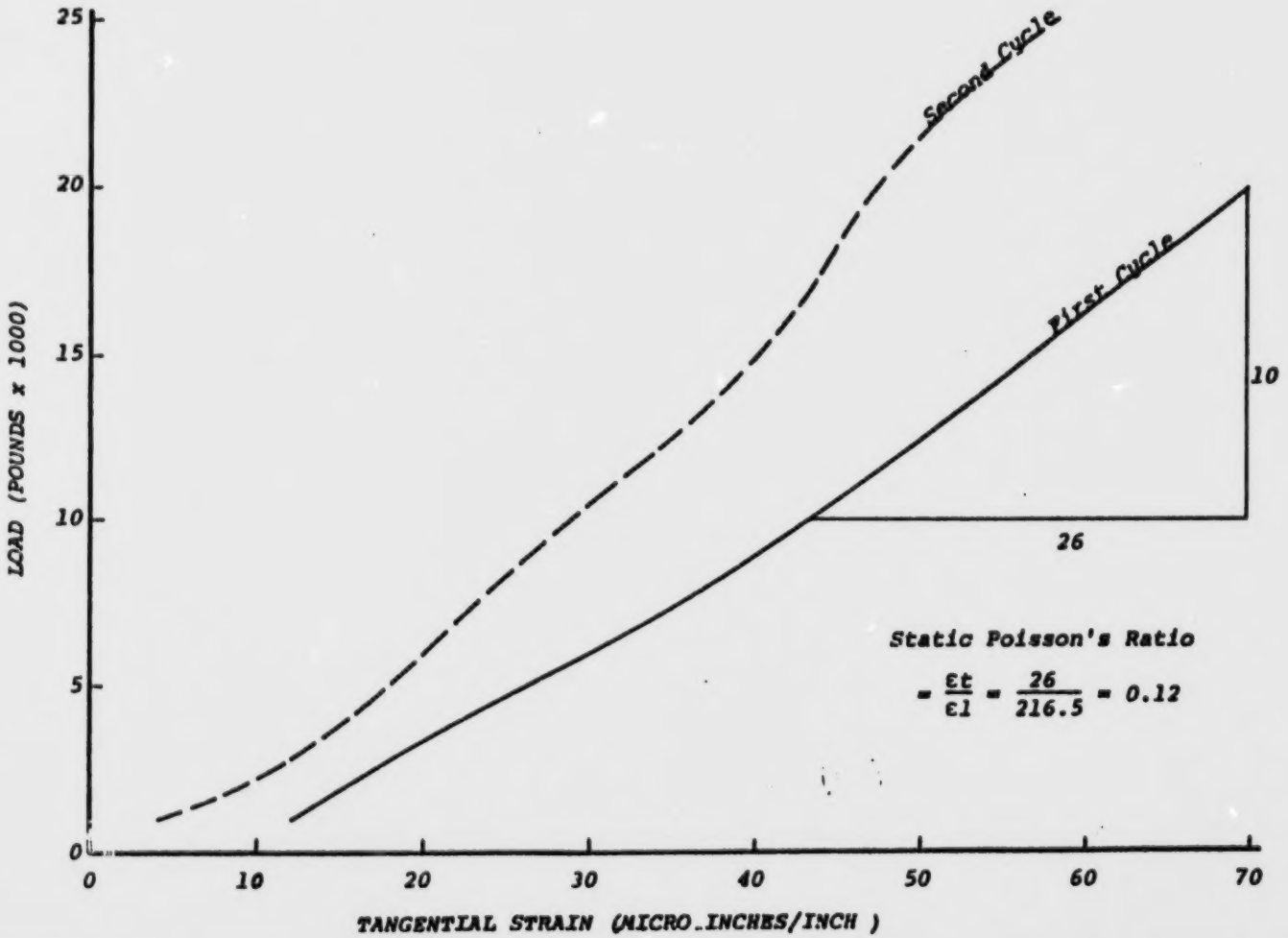
$$= \frac{50.6}{6.25 \text{ in}^2} \ \frac{\text{lbs}}{\mu\text{in/in}} = 8.1 \times 10^6 \text{ psi}$$

R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC MODULUS OF ELASTICITY

SAMPLE No. 592 SANDSTONE (medium)

SCALE as shown	DATE March/76	MADE BB	CHKD. AO	JOB: cs 1303	PLATE A-2



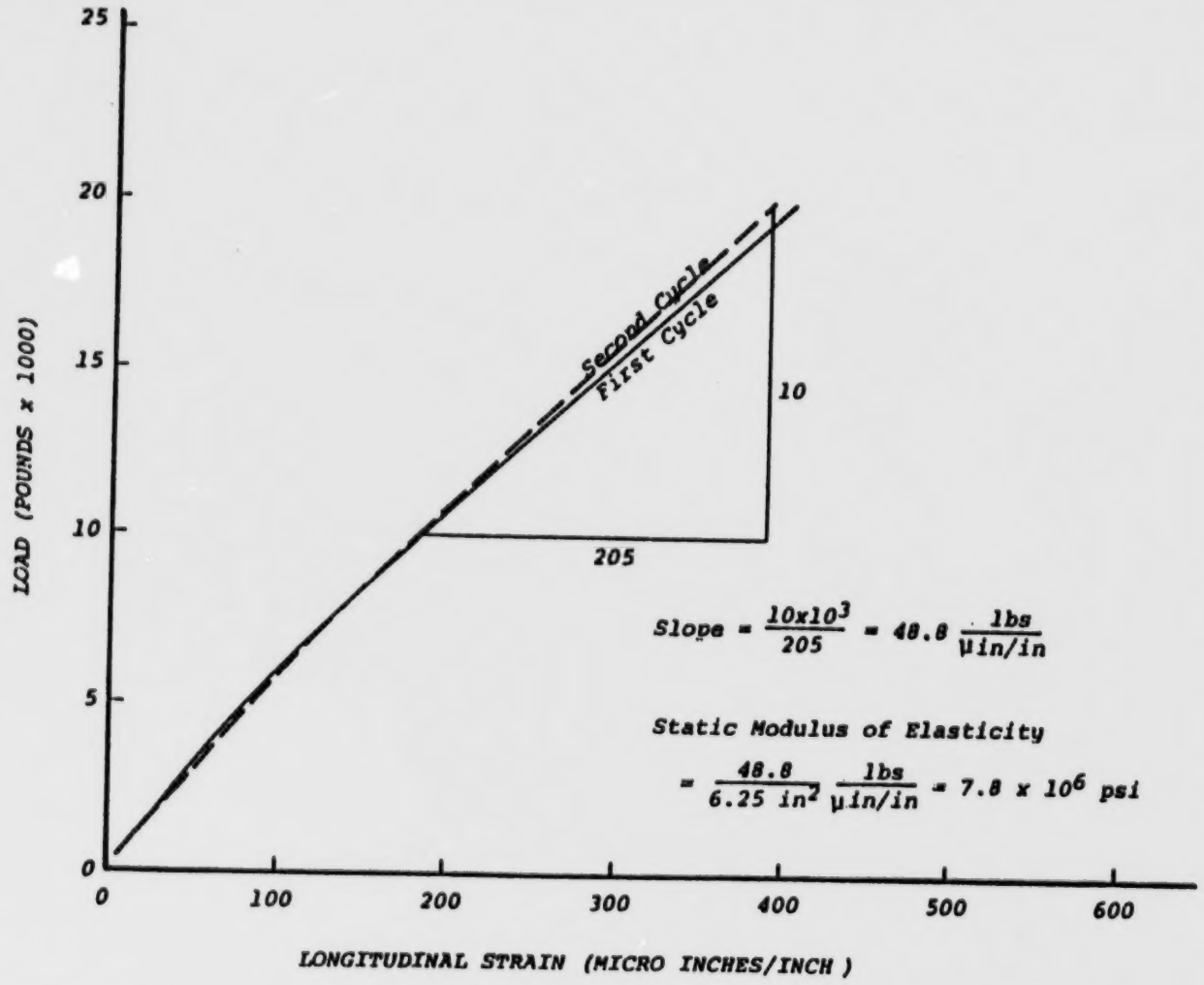
Static Poisson's Ratio

$$= \frac{\varepsilon t}{\varepsilon l} = \frac{26}{216.5} = 0.12$$

R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC
POISSON'S RATIO
SAMPLE No. 298 SANDSTONE (fine)

SCALE as shown	DATE March/76	MADE BB	CHKD. AO	JOB: GS 3303	PLATE A-5



$$\text{Slope} = \frac{10 \times 10^3}{205} = 48.8 \ \frac{\text{lbs}}{\mu\text{in/in}}$$

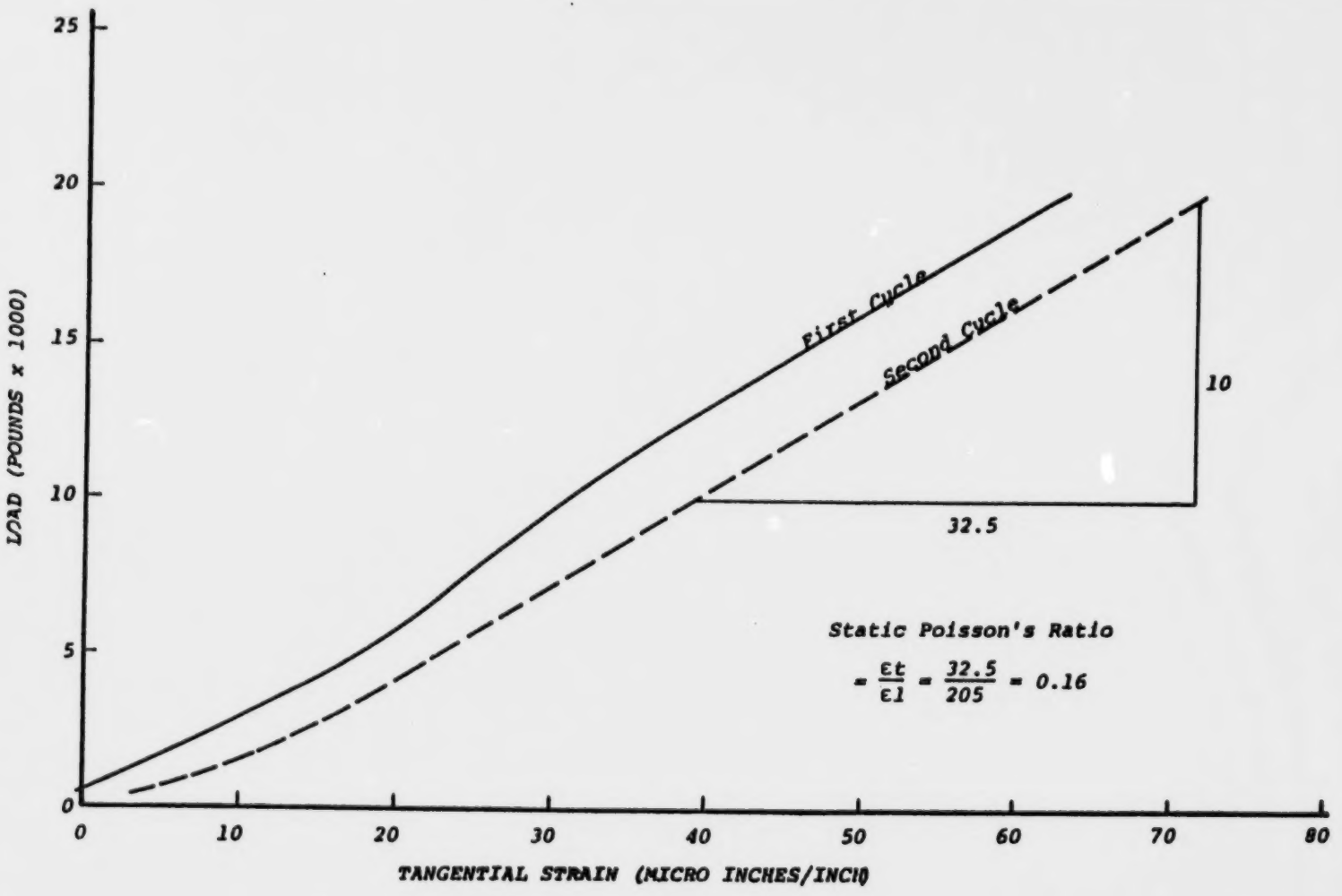
Static Modulus of Elasticity

$$= \frac{48.8}{6.25 \text{ in}^2} \ \frac{\text{lbs}}{\mu\text{in/in}} = 7.8 \times 10^6 \text{ psi}$$

R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC MODULUS OF ELASTICITY
SAMPLE No. 598 SANDSTONE (fine)

SCALE as shown	DATE March/76	MADE BB	CHKD. AO	JOB: cs 3101	PLATE A-6

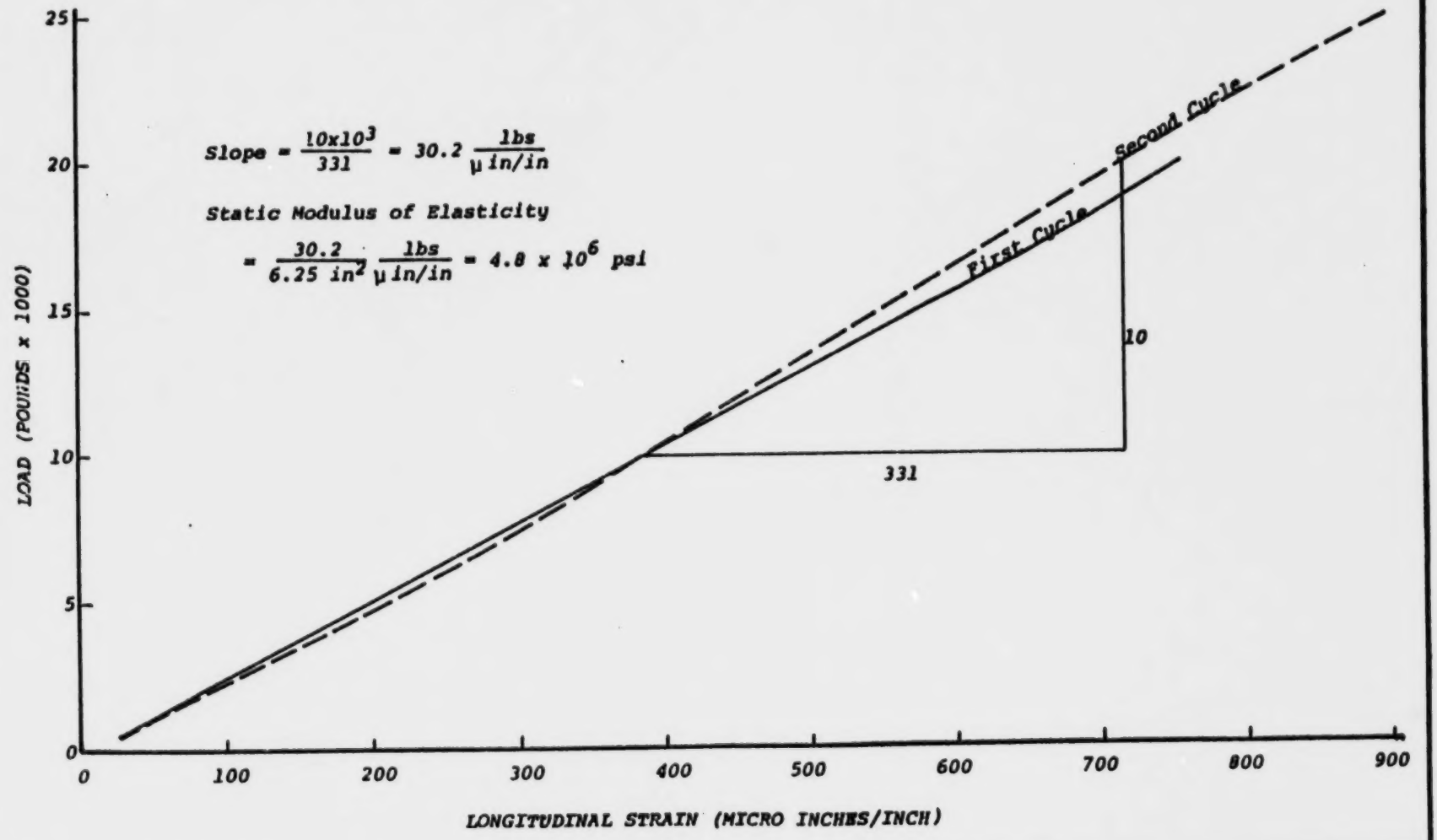


R.M. HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC POISSON'S RATIO

SAMPLE No. 598 SANDSTONE (fine)

SCALE as shown	DATE March/76	MADE BB	CHKD. AO	JOB: CS 1303	PLATE A-7

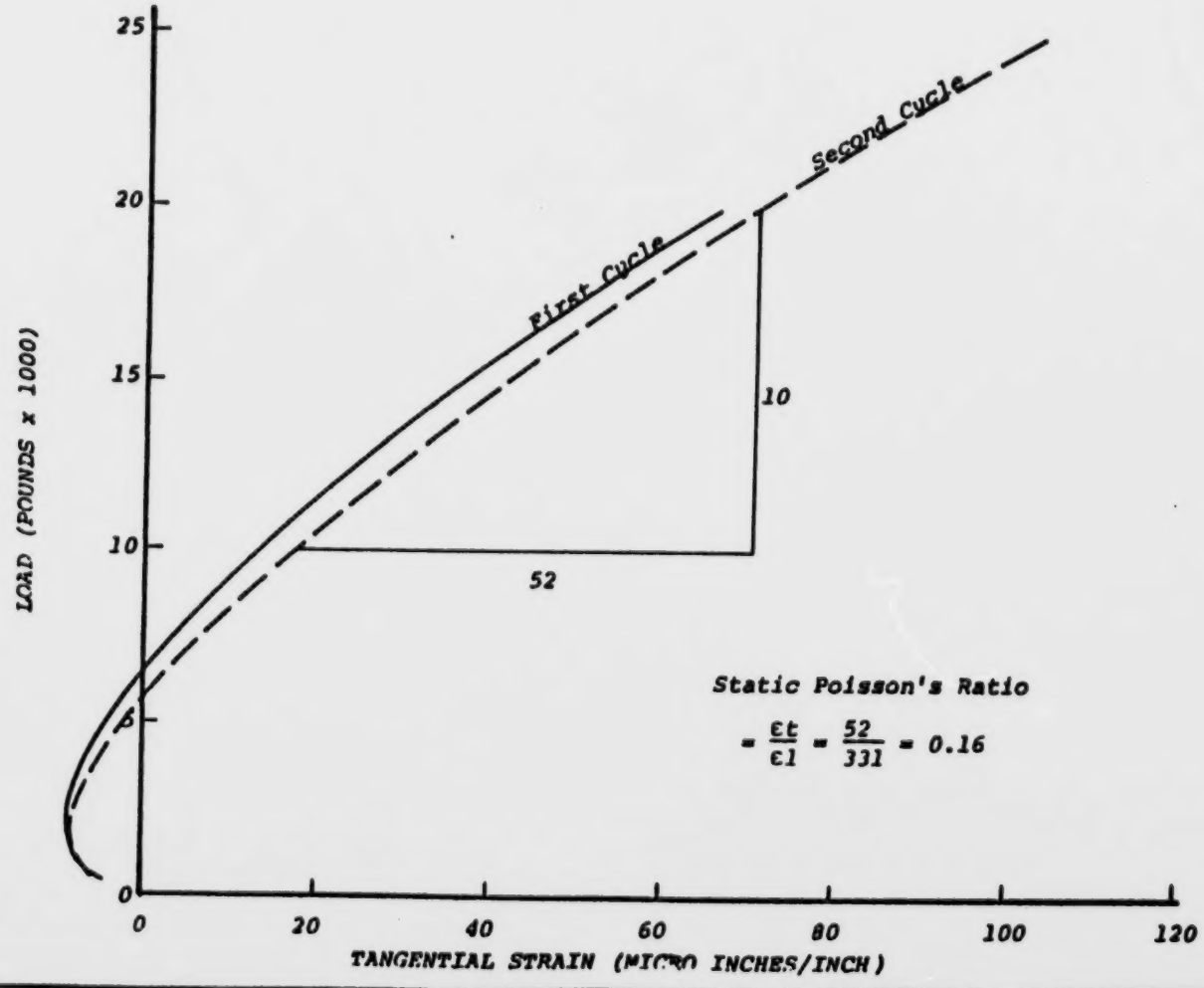


R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC MODULUS OF ELASTICITY

SAMPLE No. 288 SILTSTONE

SCALE as shown | DATE March/76 | MADE BB | CHKD. AO | JOB: cs 3303 | PLATE A-8



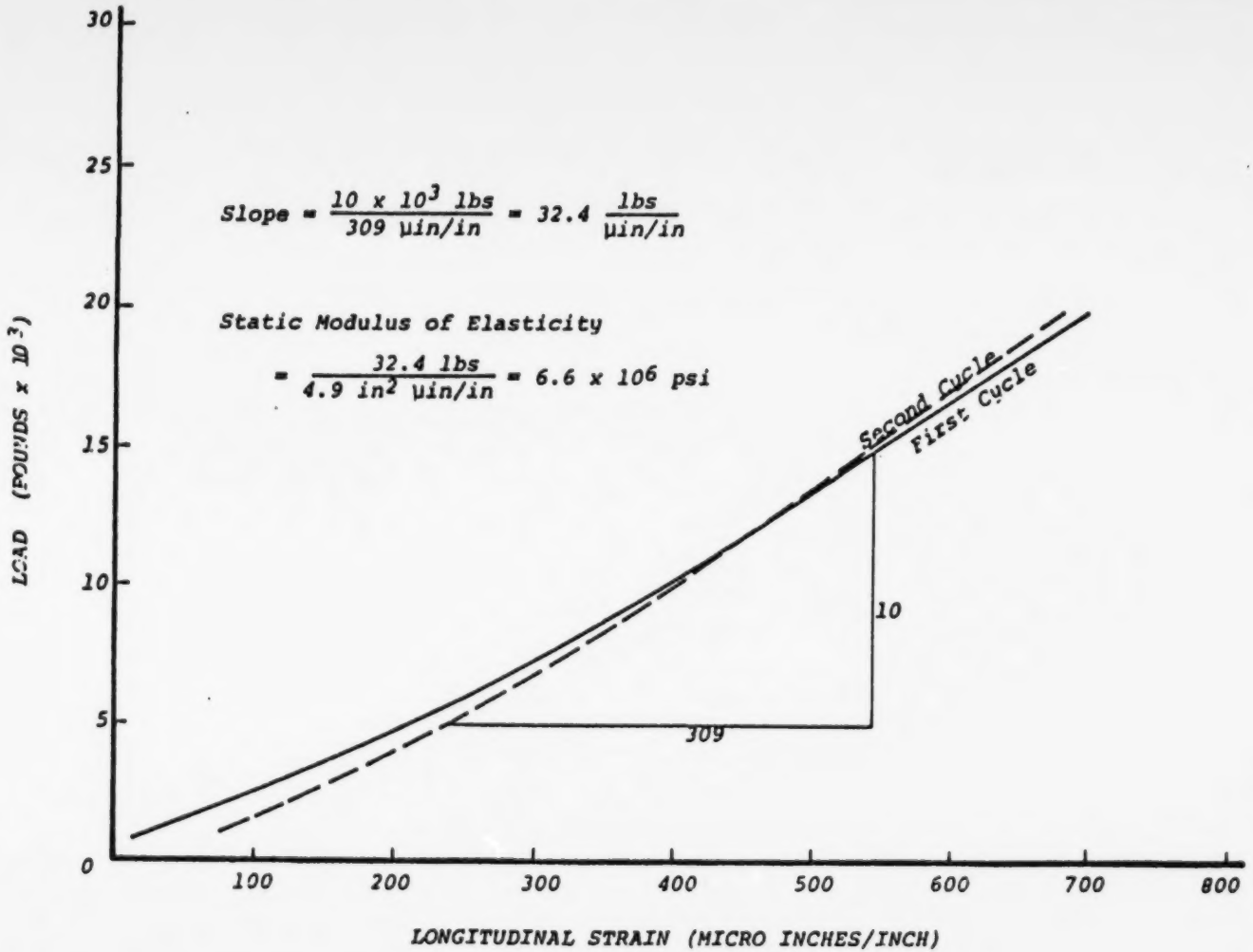
Static Poisson's Ratio

$$= \frac{\varepsilon t}{\varepsilon l} = \frac{52}{331} = 0.16$$

R.M. HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC
POISSON'S RATIO
SAMPLE No. 288 SILTSTONE

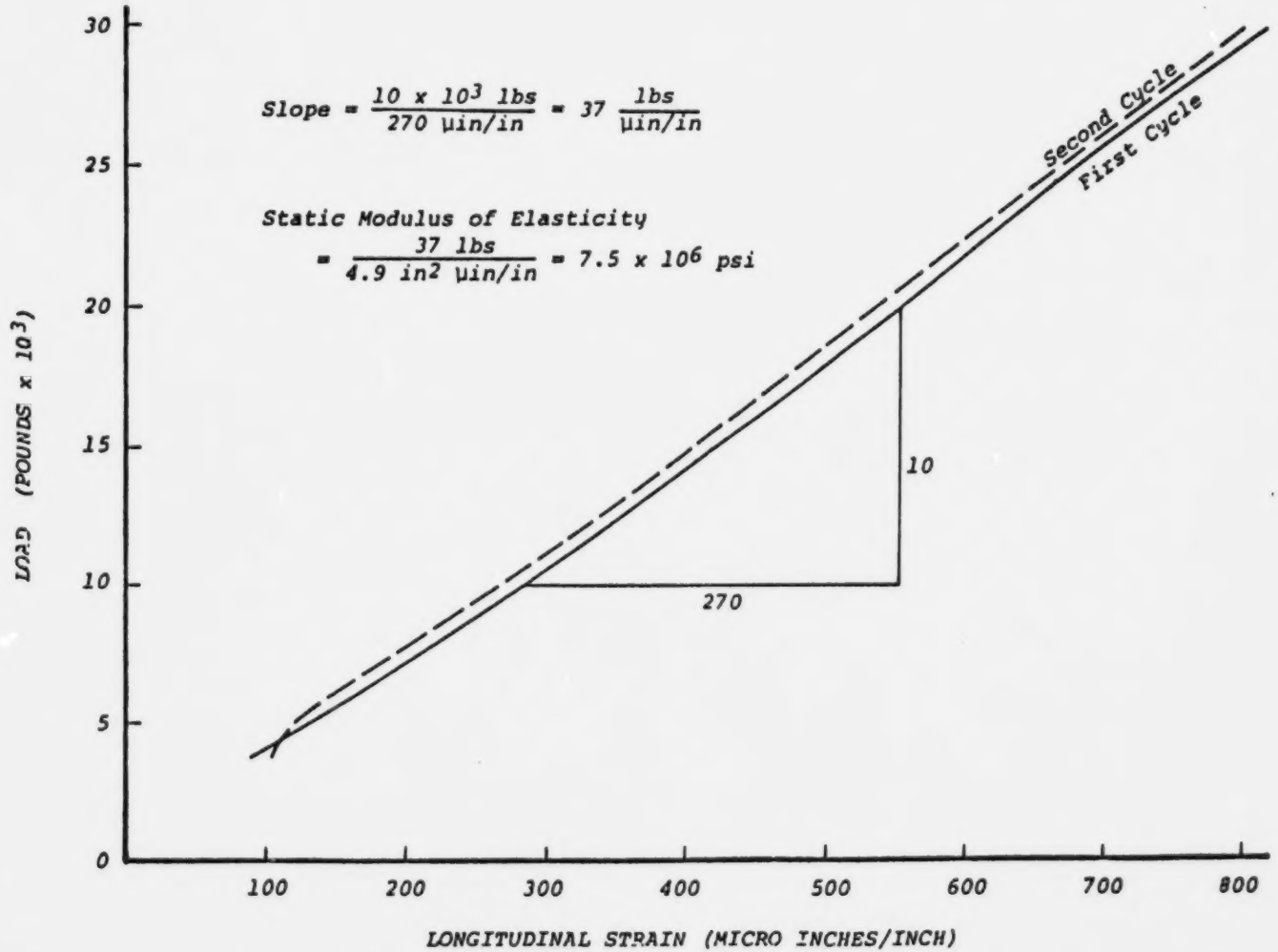
SCALE as shown	DATE March/76	MADE BB	CHKD. AO	JOB: cs 1101	PLATE A-1



R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC MODULUS OF ELASTICITY
CONFINING PRESSURE ~ 150 PSI
SAMPLE NO. 298 SANDSTONE (fine)

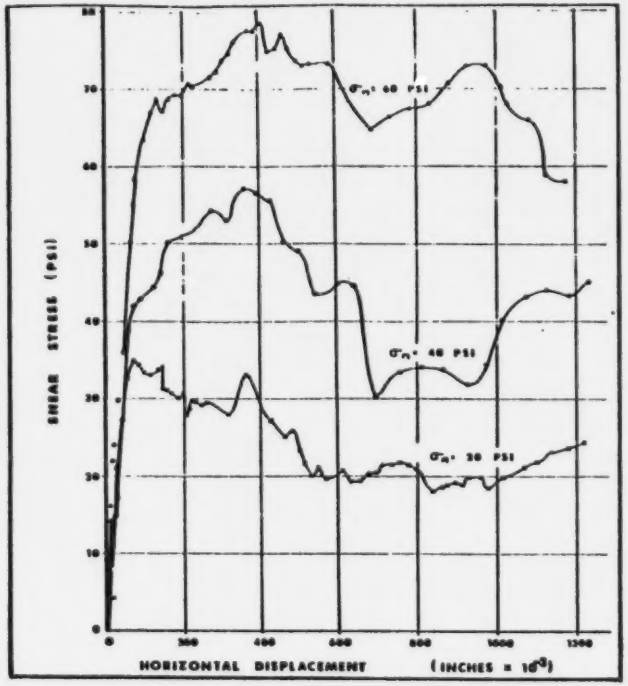
SCALE as shown | DATE March/76 | MADE BB | CHKD. AO | JOB: CS 1101 | PLATE A-10

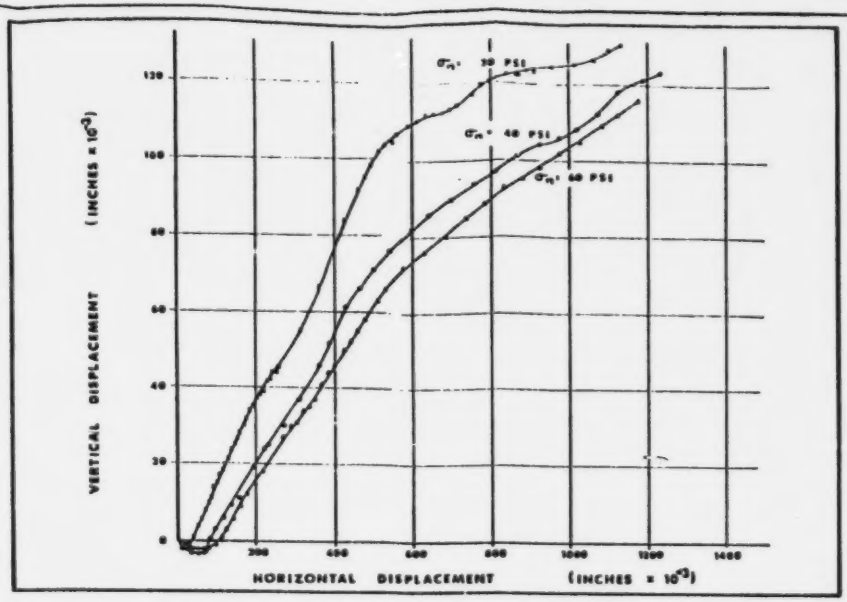


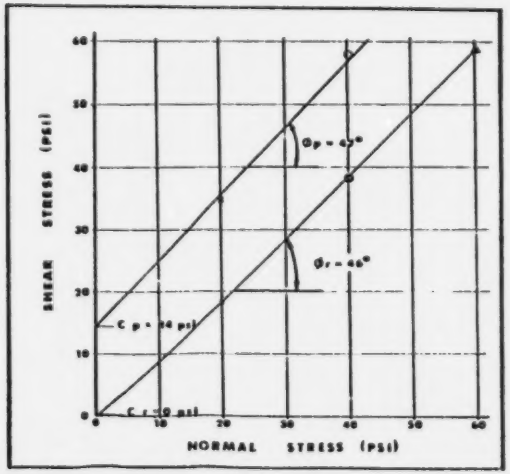
R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

DETERMINATION OF STATIC MODULUS OF ELASTICITY
CONFINING PRESSURE — 750 PSI
SAMPLE NO. 298 SANDSTONE (fine)

SCALE as shown | DATE March/76 | MADE BB | CHKD. AO | JOB: cs 1301 | PLATE A-11







NOTE:
Fresh, medium-grained, carbonaceous sandstone.
Sample broken approximately parallel to bedding.
Very rough and irregular surface.
Effective roughness angle (i) approximately 10 - 15°.
Shear rate equals 0.0033 inches/min.

No.	REVISION	DATE	BY

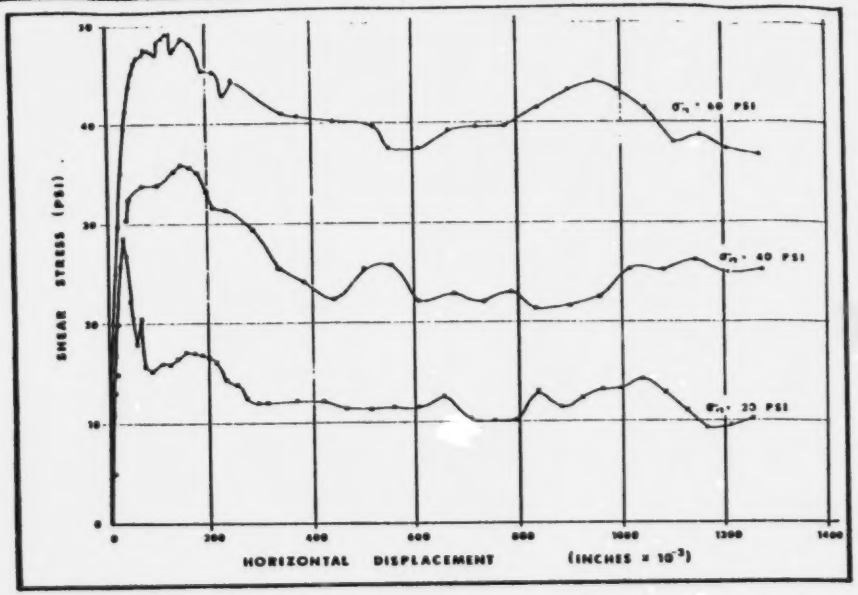
REFERENCES

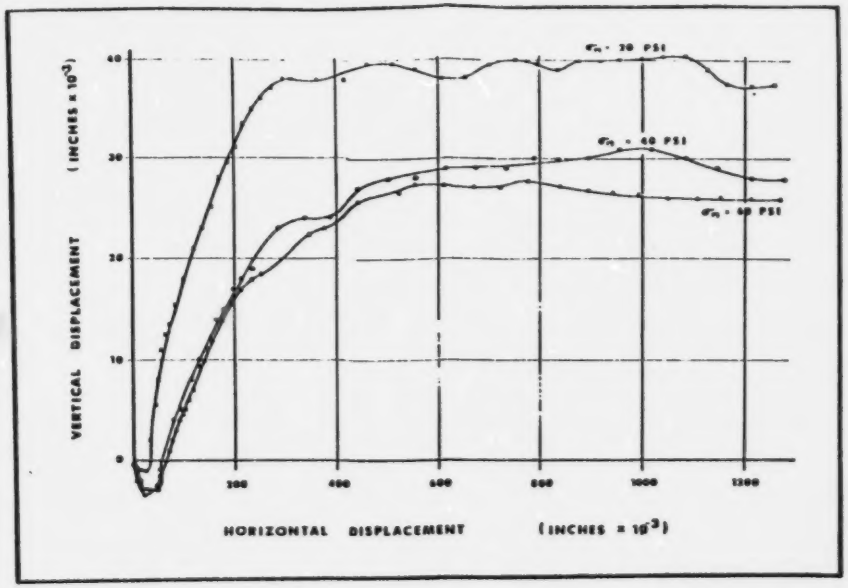
R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

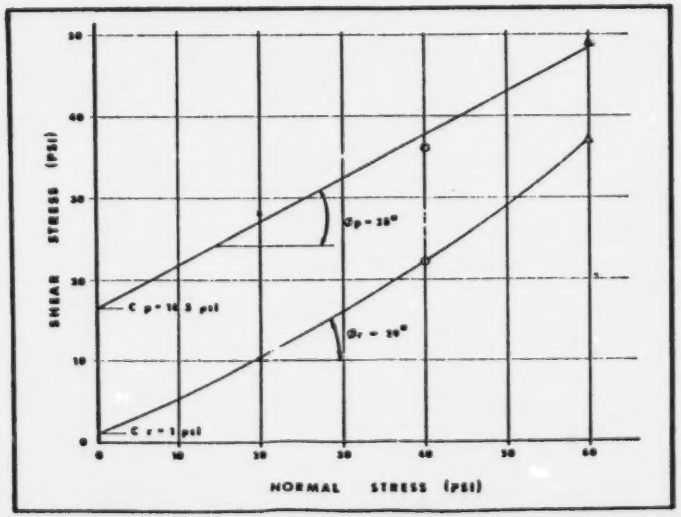
McINTYRE MINES LIMITED
NO. 9 MINE EXTENSION
DIRECT SHEAR TEST RESULTS
SAMPLE NO 298 MEDIUM-GRAINED SANDSTONE

SCALE | DATE 10/4/76 | DRAWN | CHKD | APPD.

No CS 3303 | A 12







NOTE:

Carbonaceous Mudstone.

Sample broken parallel to bedding along a shiney slickensided, coaly parting.

Undulating, rough, surface.

Effective roughness angle (i) approximately 2 - 4°.

Shear rate equals 0.0033 inches/min.

No.	REVISION	DATE	BY

REFERENCES

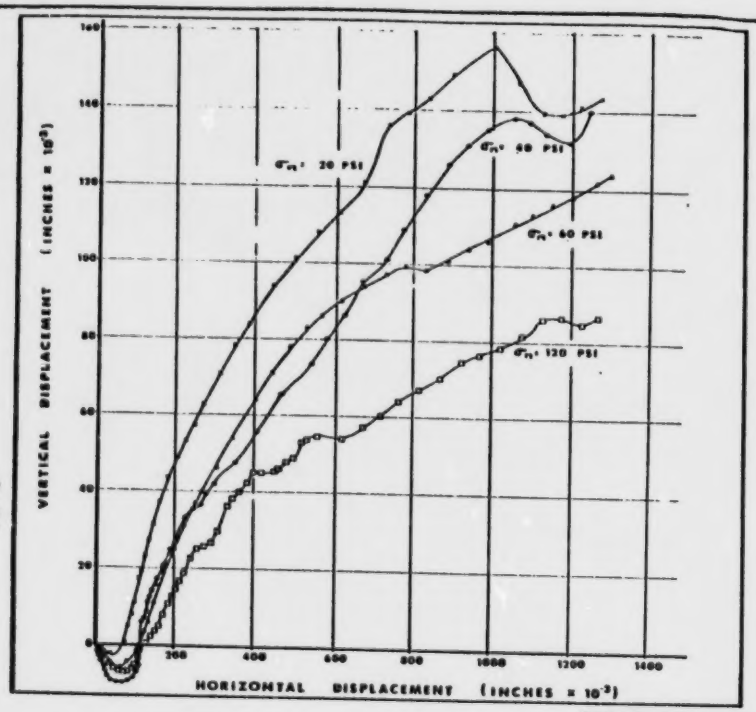
R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

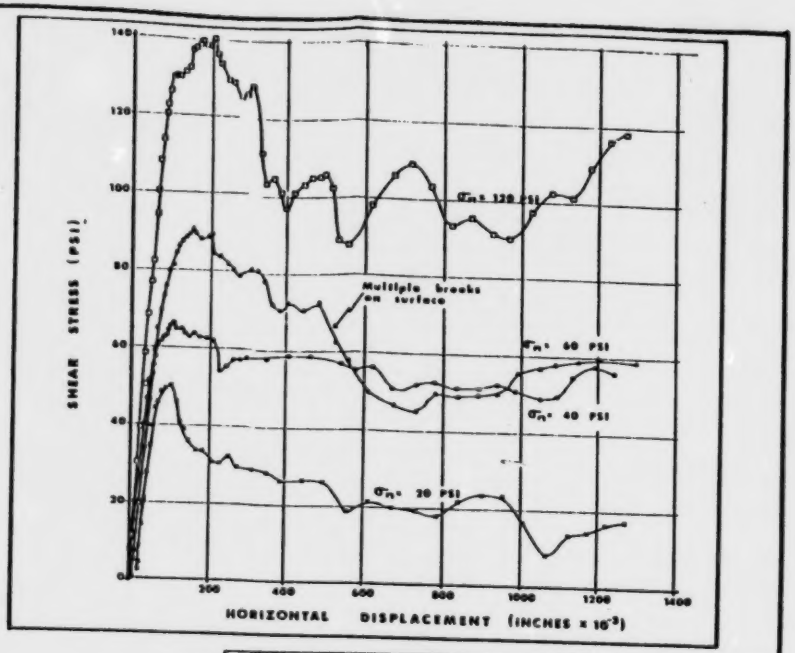
McINTYRE MINES LIMITED
NO 9 MINE EXTENSION
DIRECT SHEAR TEST RESULTS
SAMPLE NO. 394 CARBONACEOUS MUDSTONE

SCALE ... DATE 10/4/78 MADE ... CHKD ... APPD ...

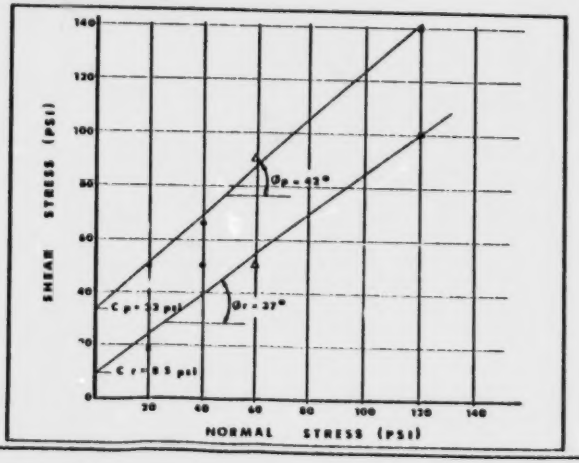
No. CS 3303 A 13

3-30





NOTE:
Finely laminated sandy siltstone.
Sample broken approximately parallel to bedding.
Rough, undulating surface.
Effective roughness angle (i) approximately 5 - 7°.
Shear rate equals 0.0033 inches/min.



No.	REVISION	DATE	BY

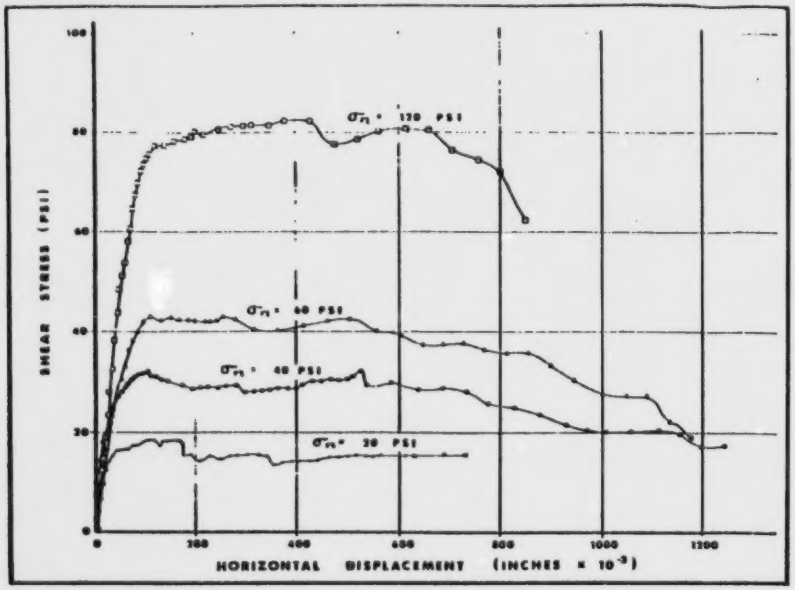
REFERENCES

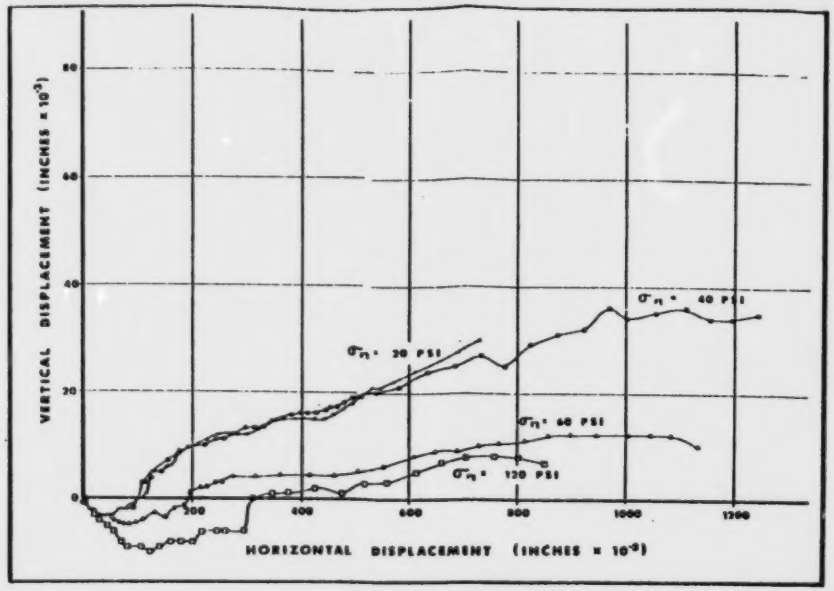
R.M.HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

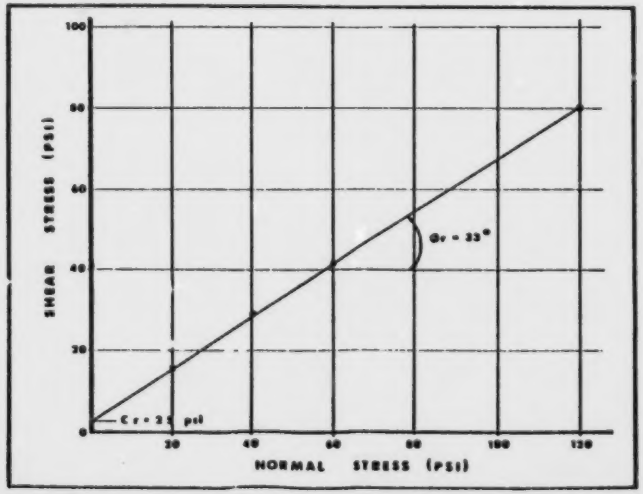
McINTYRE MINES LIMITED
NO. 9 MINE EXTENSION
DIRECT SHEAR TEST RESULTS
SAMPLE NO. 307 SILTSTONE

SCALE | DATE 10/4 76 | MADE | CHKD | APPD

No. CS 3303 | A 14







NOTE:

Badly weathered sandstone.

Sample broken parallel to bedding.

Undulating, rough, iron stained surface.

Effective roughness angle (i) approximately 5°.

Shear rate equals 0.0033 inches/min.

No.	REVISION	DATE	BY

REFERENCES

R.M. HARDY & ASSOCIATES LTD.
CONSULTING ENGINEERING & TESTING

McINTYRE MINES LIMITED
NO. 9 MINE EXTENSION
DIRECT SHEAR TEST RESULTS
SAMPLE NO S-9 SANDSTONE

SCALE DATE 10/6/76 MADE CHKD APPD.

No. CS 3303 A 15

COAL MINING RESEARCH CENTRE

"Preliminary Plate Load Tests At McIntyre Mine Grande Cache, Alberta June 1982"

BY

D.H. Green and P.L. Wright.

CMRC Report # 82/46-C JULY 15th, 1982

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A. CONTRACT

B. GRAPHS

C. FIELD NOTES

D. GEOLOGICAL SECTION - PROVIDED BY MCINTYRE

E. REPORT 82/49-C - SUPPLEMENT TO PRELIMINARY PLATE LOAD BEARING TESTS.

Preliminary Plate Load Bearing Tests at McIntyre Mines

1. INTRODUCTION

CMRC designed and had fabricated hydraulic plate load test equipment for operation under provisions of a contract with McIntyre Mines, Grande Cache for operation at 9G-4 mine during the month of June 1982. Some twenty tests were completed; these consisted of nineteen plate load tests of the floor at horizons ranging from 8.5' to 21½' from the roof and one mechanical trial test of the equipment. McIntyre prepared the site which consisted of a 550' long entry located at one end of the proposed longwall panel 402, and which had been ramped down to the sandstone footwall approx. 22' from the roof.

2. SUMMARY

Four distinct coal bands were found in No. 4 seam as determined both visually and with the plate load test equipment. Three bands are hard with yield strengths above 500 psi. One soft band 5.5' thick, located 12' through 17.5' down from the cap rock, has a yield strength of 100 psi. These yields strengths were determined through use of the plate load test equipment. These coal bands in No.4 seam at the test site are consistant with those found throughout the area.

3. EQUIPMENT

The test equipment consists principally of a 100 ton hydraulic ram and 21' of 8" dia. nominal.½" wall, 50,000 psi yield hollow round structural steel mast in flanged sections of 1', 2', 3', 4' and 6'. A bottom plate 12" x 12" is used to load the floor. The readings on the pressure guage on the hand operated hydraulic pump to convert to pressure on the 12" x 12" bottom plate at a scaling factor of 7. A supplementary report describing the equipment and the selection of plate size is attached as Appendix E.

One test was performed underground at 9G-4 mine to test the hydraulics and overall operation of the equipment. The equipment performed flawlessly.

4. TEST RESULTS

The test results are summarized in "Generalized Geological Section". 4 seam at 9G is approx. 21' thick at the test site (see Plan: 9G Mine - 4 Seam, Plan: - Test Sites; Section: 2N through test sites). This seam consists of a 11.5' band of hard coal below the cap rock; a 5.5' band of soft sheared coal, 2.5' of medium hard coal and 1.5' of hard coal above the sandstone footwall.

The locations of the plate load tests with respect to these bands were measured from the roof of the entry which lies approx. 0.5" to 0.8" into the cap rock. Four tests were made in the hard coal band below the roof, twelve tests in the soft coal, two in the medium and one in the hard band above the foot wall. The following is a summary of the results (see Summary Table or the Appendices for more detail).

# Tests	Coal Band	Thickness	Strength Range (psi) Yield	Ultimate	Total Penetration of Plate (ins.)
4	Hard	11.5'	600 and higher	900 and higher	0.23 to 2.69
12	Soft	5.5'	100 to 300	130 to approx. 400	2.33 to 6.46
2	Medium	2.5'	630	660 to above 800	0.28 to 4.83
1	Hard	1.5'	475	above 600	0.94

The loading rate on the bottom plate was usually 50 psi set every 2 minutes (see Field notes; Appendices). Several tests took longer than 2 minutes to apply 50 psi where the stress/strain modulus (slope) was very flat or where in some cases pressures beyond the yield point were reached. Ultimate strengths were determined from the graphs (see Charts: Pressure vs. Penetration), where the curves levelled off or where the maximum pumping rate was achieved at high total penetrations into the floor. In the soft coal, a penetration rate of 0.5"/minute or more were achieved.

Test (15) was set in soft coal. The medium hard coal below was sufficiently close to interfere once penetration had reached 3.50" and 425 psi. Test (13) was made a few inches deeper than test (14). Thus test (15) tested a combination of both hard and soft coal.

Test (3) was made in wet hard coal (see Field notes). Despite this, a yield strength of over 600 psi was attained.

Tests (5 and 6) were both run at 14' in soft coal. Test (5) may have had a thin band of hard coal, between bottom plate and soft coal. Results for (5) much higher than for the other soft coal tests.

The floor was scraped clear of loose and levelled before all tests. Initial readings were taken between 50 psi to 100 psi so as to eliminate the effects of the crushing of the very small pieces of coal that may remain on the flat floor. To obtain the total penetration, the slope on the test graphs were extended to intersect the pressure axis at 0 psi.

Tests (18) and (19) were stopped at 600 psi and 800 psi respectively in order to protect the equipment which was extended to 21' and 19' respectively. By comparing the modulus (slope) of the pressure - penetration (stress-strain) relationship of test (19) and others, one can determine that test (19) has a yield strength beyond 1000 psi. Test(18) has a yield strength of 475 psi.

Overnight and over the weekend tests were made which were outside of the terms of the contract. Penetrations of no more than 0.15" were obtained over a two day period. Final pressures were close to the yield points.

The plate load tests gave, in the opinion of the authors direct readings on the yield and ultimate strengths of the coal bands tested.

RECOMMENDATIONS

It is recommended that further test sites be carefully considered for plate load tests along 1 west beside 402 panel after this heading is further advanced to completion.

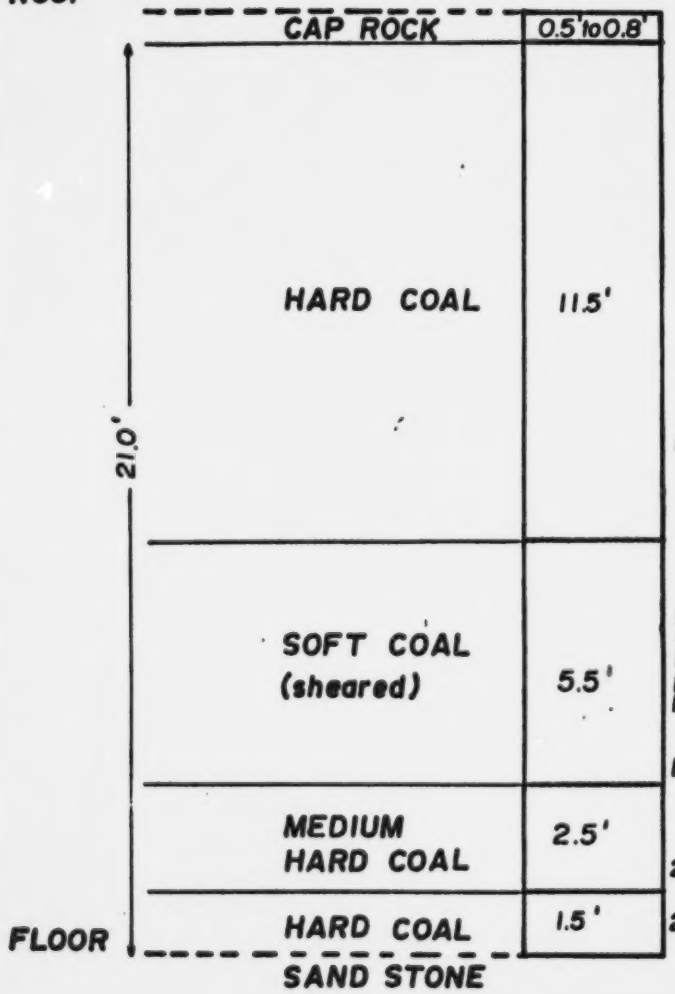
It is recommended that the penetration tests, as recommended by CMRC and which are in progress in conjunction with McIntyre's core drilling program in 9G-4, be continued.

D.H. Green, P.Eng,
Senior Project Engineer.

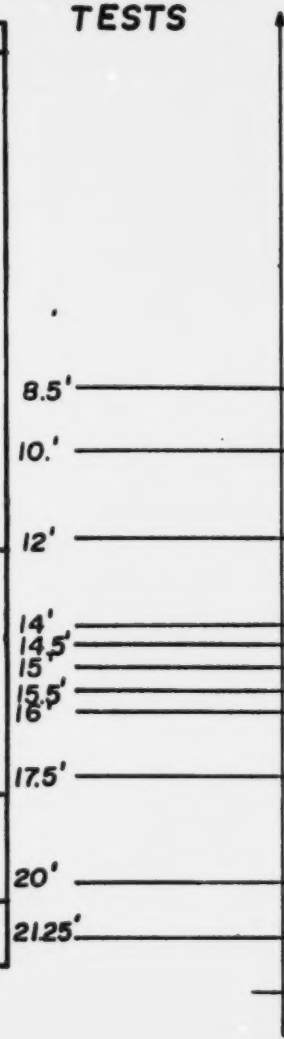
P.L. Wright. P.Eng,
Manager, Mining Technology Division.

UMRC

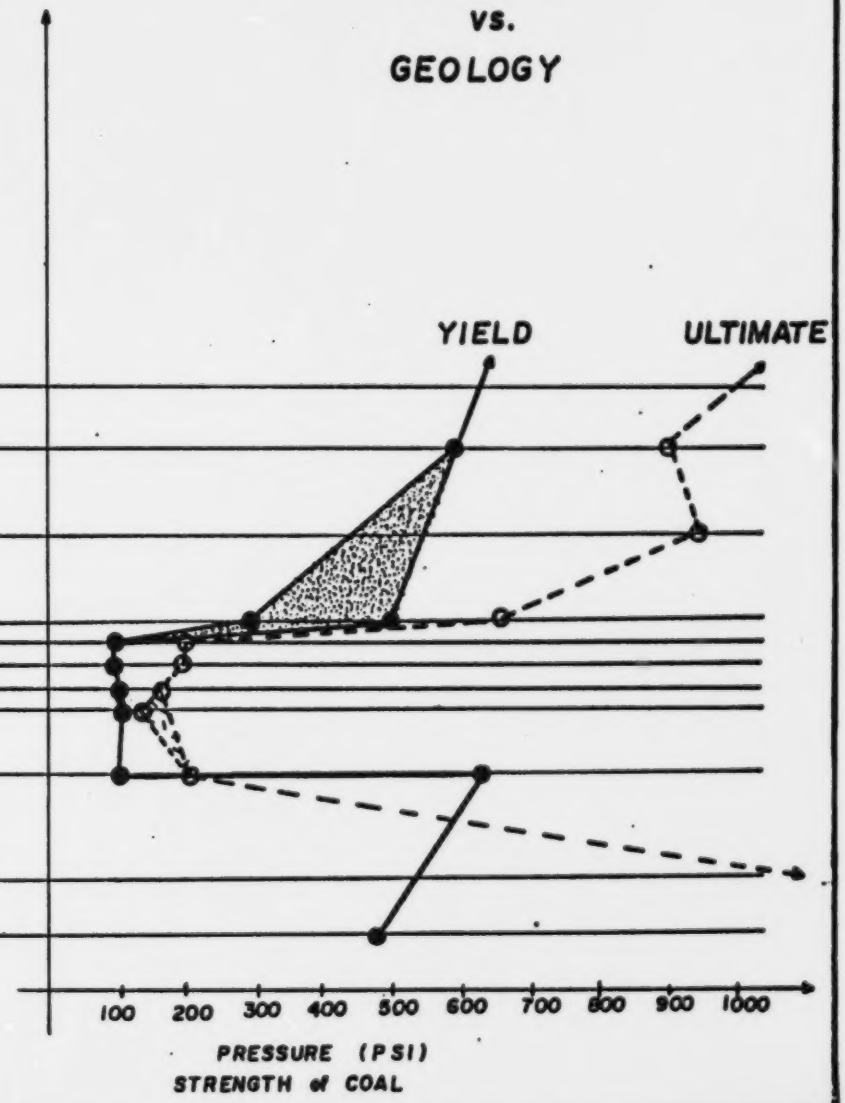
GENERALIZED GEOLOGICAL SECTION

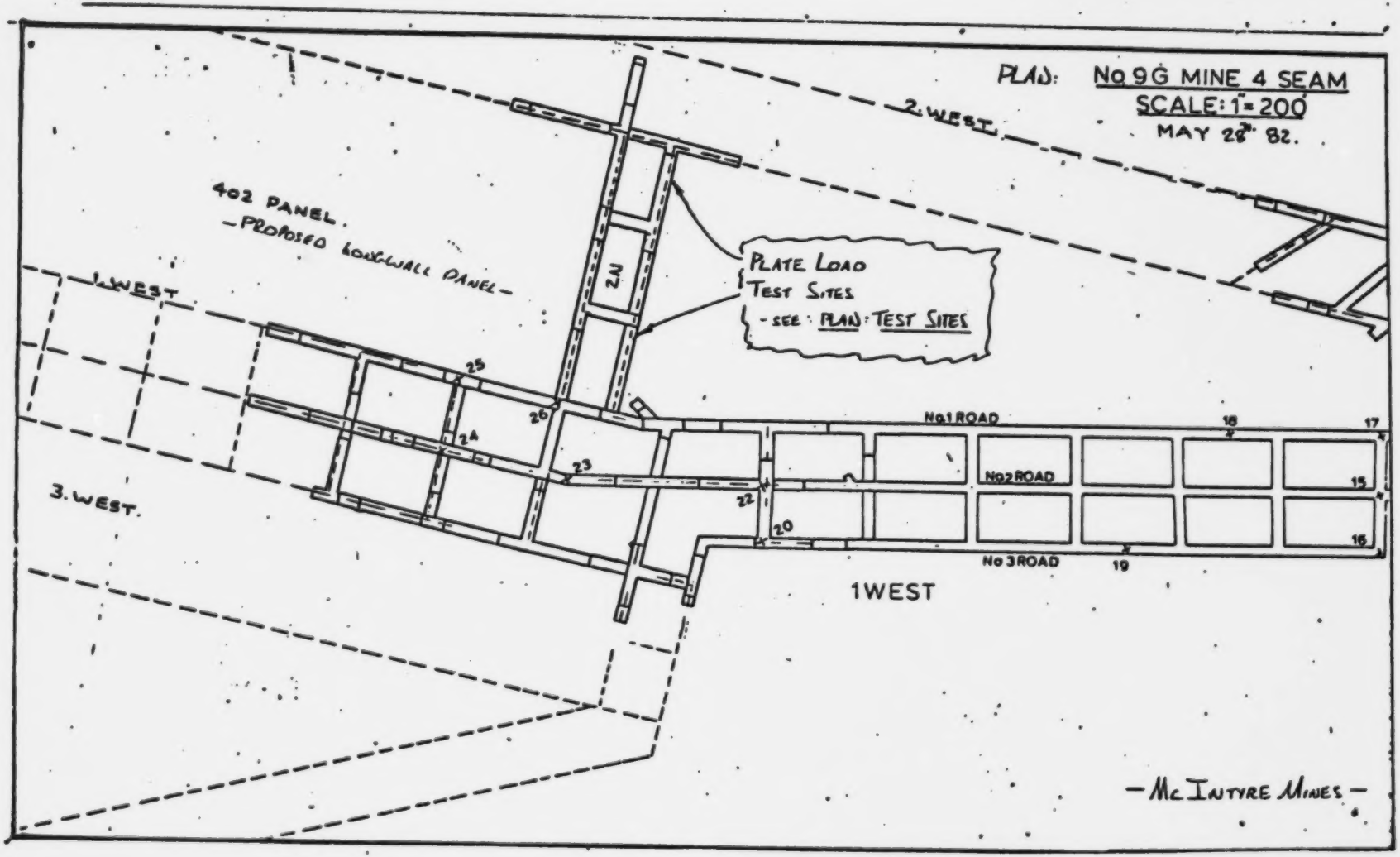


LOCATION of PLATE LOAD TESTS

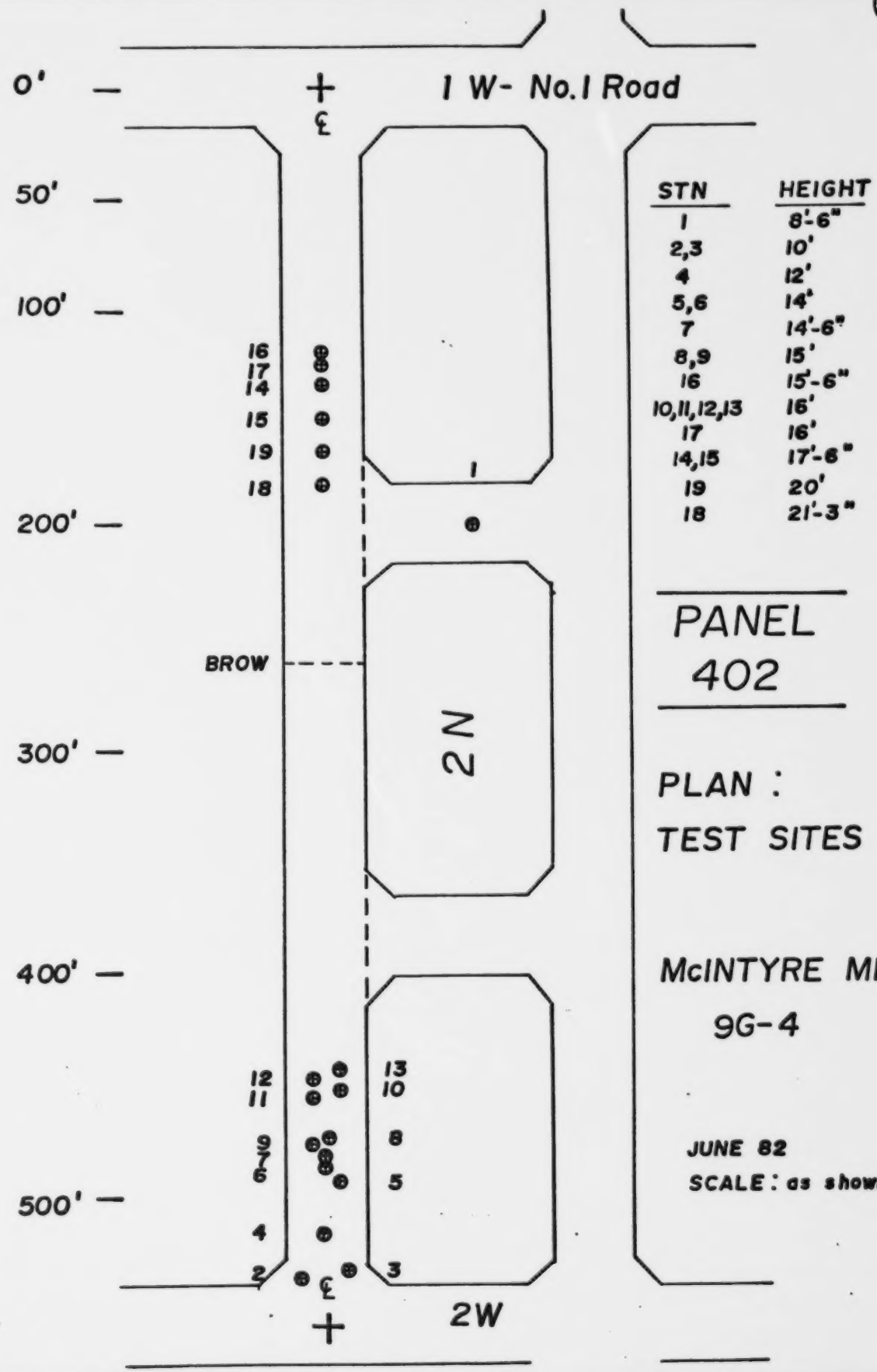


GRAPH : YIELD/ULTIMATE STRENGTH vs. GEOLOGY




PLAN: No 9G MINE 4 SEAM
SCALE: 1" = 200'
MAY 28th 82.
2. WEST.
402 PANEL.
- PROPOSED LONGWALL PANEL -
1. WEST
2N
PLATE LOAD
TEST SITES
- SEE: PLAN: TEST SITES
25
26
24
23
22
20
No1 ROAD
No2 ROAD
No3 ROAD
18
17
15
16
19
3. WEST.
1WEST
- McINTYRE MINES -

CMRC



STN	HEIGHT
1	8'-6"
2,3	10'
4	12'
5,6	14'
7	14'-6"
8,9	15'
16	15'-6"
10,11,12,13	16'
17	16'
14,15	17'-6"
19	20'
18	21'-3"

PANEL 402

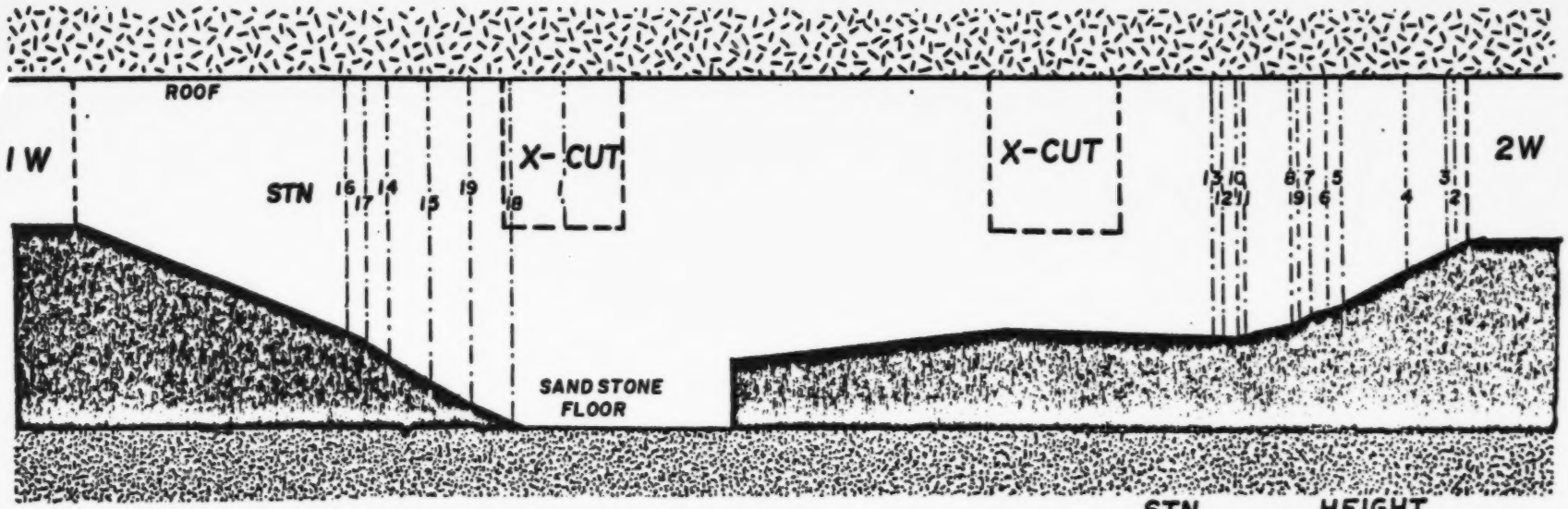
PLAN :
TEST SITES

McINTYRE MINES
9G-4

JUNE 82
SCALE : as shown

CMRC

SECTION: 2N through TEST SITES



SCALE: VERTICAL 1" = 8'
HORIZONTAL 1" = 60'

NOTE: LOCATION OF TEST SITES ARE SHOWN AS VERTICAL LINES
SLOPE OF STRATA DOWN FROM 1W to 2W is NOT INDICATED

STN	HEIGHT
1	8'6"
2,3	10'
4	12'
5,6	14'
7	14'6"
8,9	15'
16	15'6"
10,11,12,13	16'
17	16'
14,15	17'6"
19	20'
18	21'3"

SUMMARY TABLE

Stn.	HEIGHT	COAL BAND	PRESSURE (STRESS) ULTIMATE	YIELD	TOTAL	TOTAL	PENETRATION (STRAIN) MODULUS PSI/IN.
							0 psi. 500 psi 1000 psi
1	8.5'	Hard	-	-	1000	0.23	0 - 1000 psi; 4000
2	10'	Hard	-	-	1000	1.09"	750; 0 - 500 psi.; 1135; 500 - 1000 psi
3	10'	Hard	900	600	900	2.81"	475; 0 - 600 psi
4	12'	Hard	-	-	950	2.69"	0-200 psi; 1150; 1600; 200-650; 1800; 650-950
5	14'	Hard/Soft	650	500	650	2.95"	300; 0-500
6	14'	Soft	-	300	400	3.75"	0-100; 125; 90; 100-400
7	14.5'	Soft	200	100	200	3.12"	0-100; 130
8	15'	Soft	185	100	185	3.82"	0-100; 115
9	15'	Soft	180	100	180	2.33"	0-100; 140
16	15.5'	Soft	160	-	160	6.46"	25; 0-160
10	16'	Soft	160	100	160	2.41"	95; 0-100; 100-160; 40
11	16'	Soft	165	100	165	3.27"	80; 0-100
12	16'	Soft	170	100	170	3.60	85; 0-100
13	16'	Soft	160	145	160	2.46"	80; 0-145
17	16'	Soft	130	115	130	5.05"	40; 0-100
14	17.5'	Soft	215	100	215	4.58"	0-100; 140; 100-170; 40
15	17.5'	Medium	-	630	660	4.83"	100; 0-425; 425-575; 325
19	20'	Medium	-	-	800+stopped	0.28"	1800; 0-300; 4200; 300 - 800+
18	21.25'	Hard	-	475	600+stopped	0.94"	525; 0-300; 300-475; 870

CONTRACT BETWEEN McIntyre Mines, Grande Cache, Alberta, and Coal Mining Research Centre to govern:

PROJECT: Preliminary Plate Load Bearing Tests at McIntyre Mines, No.9G-4 Mine.

TABLE OF CONTENTS

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Scope	1.0
CMRC agrees to	2.0
McIntyre agrees to	3.0
Test Site Location and Design	4.0
Schedule of Work	5.0
Operating & Report Costs of Test	6.0
Operating Procedures	7.0
Terms	8.0
Charts and Maps	9.0

PROJECT: Preliminary Plate Load Bearing Tests at McIntyre Mines Ltd. No. 9G-4 Mine.

1.0 SCOPE: This contract covers the operation of a plate load test equipment at McIntyre Mines Ltd. by the Coal Mining Research Centre for purposes of determining the compressive strength of mine floor in No. 4 seam at No. 9G-4 Mine.

2.0 CMRC AGREES TO:

1) Provide supervision and operation of the plate load test equipment through provision of one Project Engineer and one Technician on site chargeable to McIntyre Mines Ltd. at standard rates (see Schedule of Rates - Section 6.0)

2) Provide surface transportation to and from Grande Cache chargeable at regular rates.

3) Provide a confidential report to McIntyre Mines which assesses the bearing strength of the horizons tested in regards to the practicality of establishing a longwall mining system in No. 4 seam. Recommendations will also be made as to the advisability of conducting a second phase of plate loading tests.

4) Co-operate with McIntyre Engineers in the evaluation of all test data and in the choice of test sites.

5) Comply with McIntyre safety procedures and statutory management instructions while on mine property.

3.0 MCINTYRE AGREES TO:

1) Provide transportation at the mine site of materials and equipment to and between test sites.

2) Provide the service of a L.H.D. to support the test project through out its duration.

3) Provide statutory supervision at the working place.

4) Provide safety cables and anchoring points for stabilizing the test equipment as required.

5) Release all data from the tests to CMRC for compilation of a Confidential Report.

6) Pay for the services of a CMRC Project Engineer and Technician

at standard rates (see Schedule of Rates) on, to and from the mine site for the duration of the project and for food, accommodation and transportation of same also at standard rates (see Section 6).

7) Supply test sites underground in accordance with agreed to design and location in section: Test Site Location and Design.

8) Provide timber to wedge cylinder between floor and roof: this to consist of fir and a minimum of 4"x 6" sq. x 24" timber, 1" and 2" thick boards and wedges.

9) Provide underground surveying level or transit c/w tripod and a rod which reads to within 0.01 in. or 1 mm.

4.0 TEST SITE LOCATION AND DESIGN

Test sites are required according to the following specifications:

1) Designate cross-cuts or drivages as test sites and pre-cut floor in all but one of the cross-cuts in advance of the arrival of test equipment - see Plan: Test Site Locations

2) A minimum of six cross-cuts are required. The roof for five shall be at the normal position at a 10' nominal height above the floor, with floor cutting required at depths of 0', 2', 4', and 6' - see sketch: Test Site Design I, for four of them.

   It is anticipated that at one depth of floor cut, pressure readings below 600 psi will be found. A repetition of this floor cut would then be required at a test site near the face of this depth of cut.

   A sixth cross-cut is requried ramping down to 8' below the normal floor, while maintaining the nominal 10' high roof - see sketch: Test Site Design II. A level area some 15' min. to 20' long is required at the 8' level. The ramp must be safely bolted and timbered.

3) Each test site must be bolted on a 4' x 4' spacing. All bolts must have plates with holes in one corner suitable for attaching safety cables and fittings to, to secure plate load cylinder. (McIntyre to supply cable and fittings: 5/8" ϕ messenger cable and crosby clamps or equivalent).

4) Each test site requires from one to five plate load tests depending on the load in psi determined in the first test. The following number of tests are required for each load range.

Load Range on Foot Plate	Number of tests
> 1000 psi	1
750 to 1000 psi	2
600 to 750 psi	3
500 to 600 psi	4
500 psi or less	5

NOTE: 500 psi is likely to be the critical pressure - maximum operating pressure on bottom of long wall shields of 418 psi and 80 psi of safety. Ref: Dowty 4/625 shields.

An additional test is required as a trial run to test out the equipment and operating procedures.

It is anticipated that some 20 plate load tests shall be required during this preliminary phase of testing.

5.0 SCHEDULE OF WORK

1 day - to haul equipment underground, set up and test equipment

5 days - to run 20 tests of 4 tests per day

2 days - travelling time Edmonton to Grande Cache, and cleanup and removal of equipment from underground

2 days - write report

10 days

6.0 OPERATING AND REPORT COSTS OF TEST

Professional Services (Standard Rates)	10 days x Senior Project Engineer @ $425 per day.	$4,250
	8 days x Technologist @ $250 per day.	$2,000
Accommodation - meals 7 days x 2 x $35 per day		420
- room 7 days x $40 per day		280
Transportation - Vehicle rental 8 days @ $160 + 12¢ per kilometer		400
- Gas and Oil		200
Miscellaneous - Supplies, phone.		200
TOTAL		$7,750

The above charges are estimates and which will not be exceeded

provided the project stays within scope and the number of tests do not exceed 20. Actual charges will be invoiced expenses plus fees for professional services. Any delays in excess of ½ day as created by McIntyre Mines while on site shall be charged additional. McIntyre Mines may provide technician in substitution for CMRC technician.

7.0 OPERATING PROCEDURES

1) The erection of the plate load test equipment shall be made with two CMRC employees plus a scoop and operator of McIntyre's

2) The erected extension cylinder shall be secured to posts or rock bolts with cable.

3) The operation, of the hydraulics and the reading of the instruments (Pressure gauge and surveying instrument) shall be by CMRC employees.

4) The loading of the floor by means of the plate load test equipment shall be according to the following procedures:

   a) assemble all equipment in place on test site, load to 100 psi to test for leaking hydraulic lines.

   b) preload floor to 300 psi on 12" x 12" bottom plate (not gauge reading). Increase load on base in increments of 50 psi every two minutes until failure or until 1000 psi loading is reached See chart: Mechanism of Loading measure deflection in floor with surveying instrument at mid point in two minute cycle. Record deflection and pressure readings. Failure is defined as that pressure which loads the floor to its ultimate strength see Chart: Typical Stress Strain Curve, or a deflection of 6".

   c) Test sites are to be no closer than 5' apart. Floor is to be chiseled level to fit the 12" x 12" bottom plate.

   d) Note: One foot wide foot has a depth of influence of approx. 2 feet. This approximates the 2' floor cut interval for the test sites.

8.0 TERMS

(1) CMRC will place purchase order for equipment as soon as the contract is signed.

(2) Signed contract should be accompanied by cheque for 50% ($3,875) of estimated costs. Remaining costs will be billed on completion of report.

(3) McIntyre Mines Ltd. or CMRC may cancel the contract, in writing, at any time. In that event CMRC shall immediately cease to charge for the project, and will return to McIntyre any unexpended portion of the advance, having deducted costs/committments made to the time of cancellation.

SIGNING AUTHORITY FOR
McIntyre Mines

_____

TITLE: _____

DATE: _____

_____

TITLE: _____

DATE: _____

SIGNING AUTHORITY FOR
Coal Mining Research Centre

_____

TITLE: _____

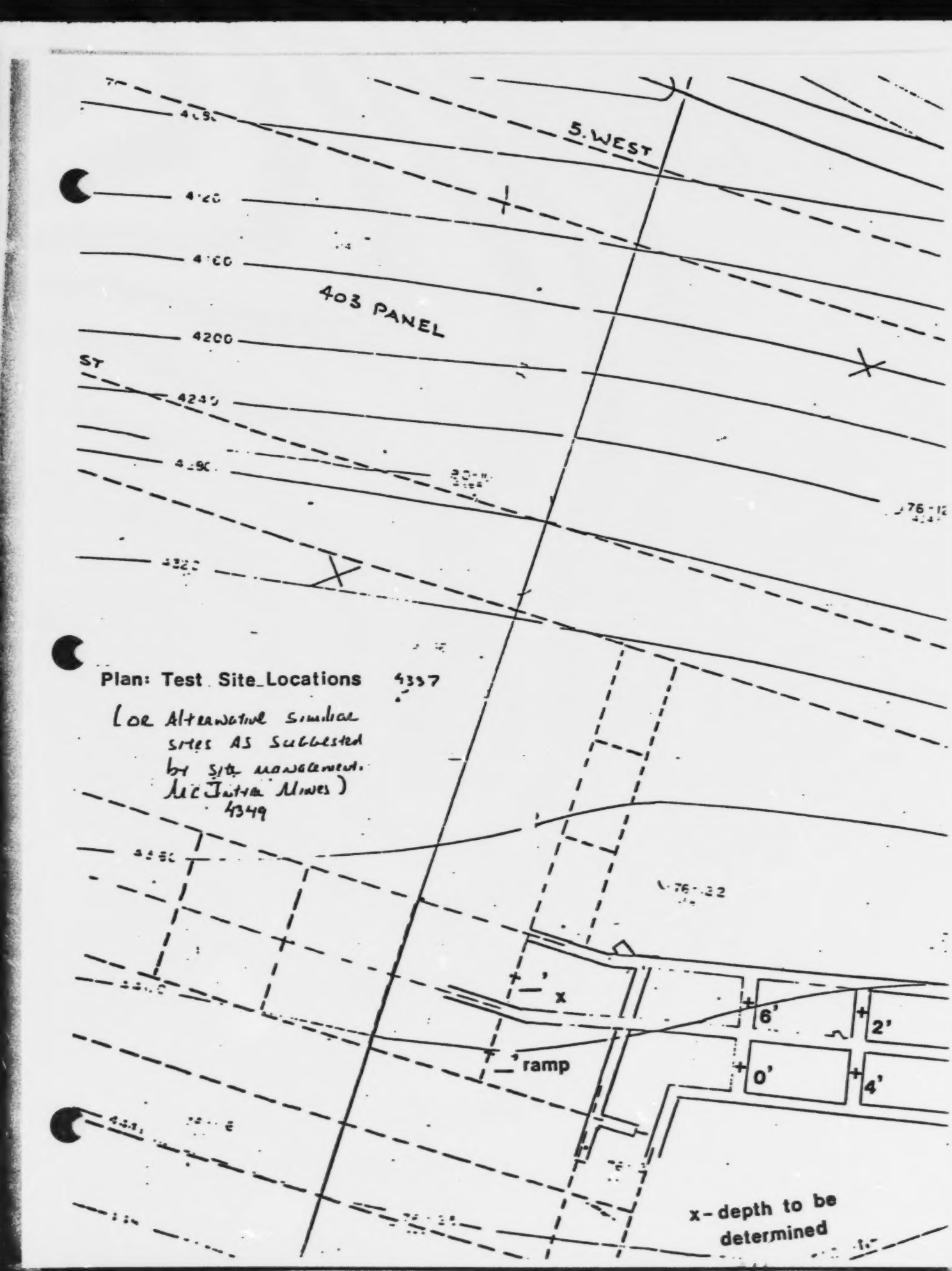
DATE: _____

_____

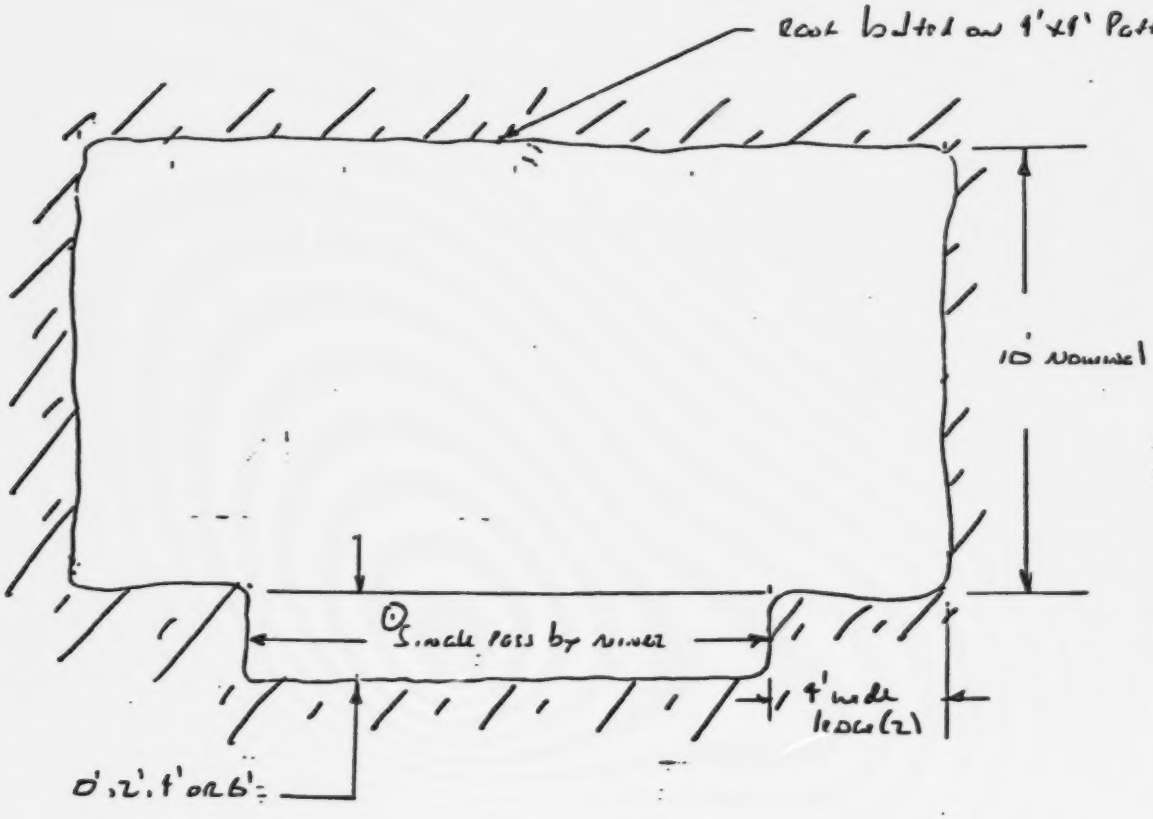
TITLE: _____

DATE: _____

9.0 CHARTS AND MAPS


5. WEST
403 PANEL
4200
ST
4339
4349
Plan: Test Site Locations
(OR Alternative similar sites AS Suggested by Site management. McIntyre Mines)
x
ramp
6'
2'
0'
4'
x - depth to be determined

SKETCH: Test Site Design I: 0' to 6' Floor Cut
- Loads in X-cut

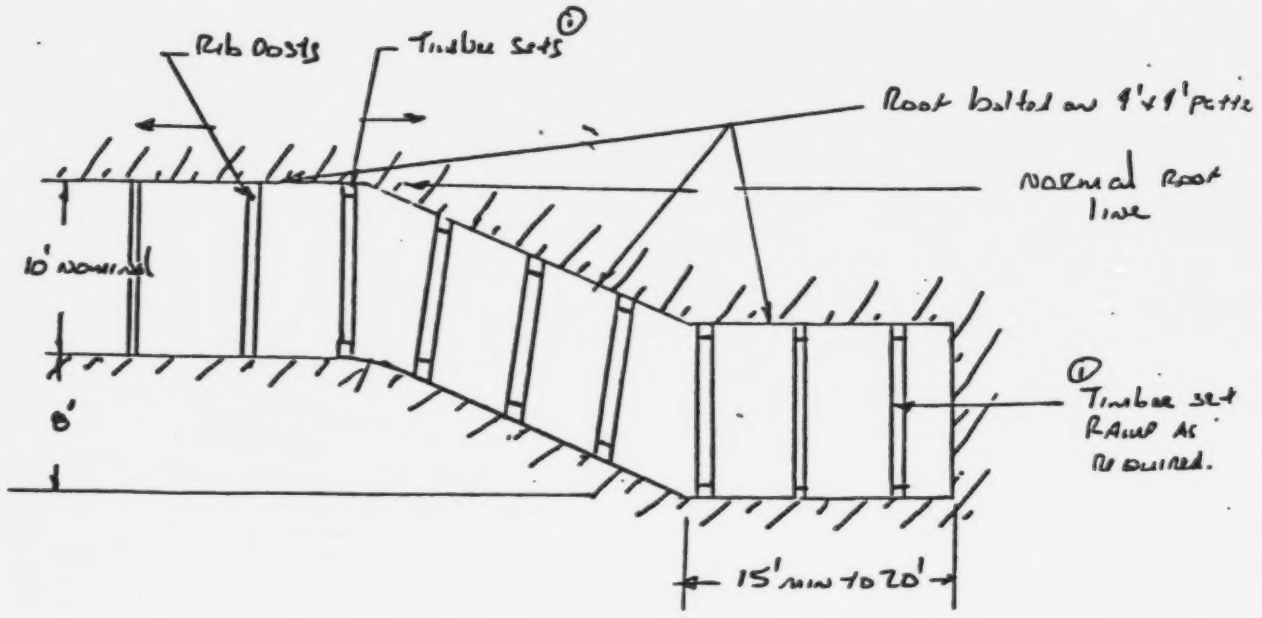


Notch level location
Floor Cutting - 15' min. to 20'.

Scale: NTS

SKETCH . TEST SITE DESIGN II : RAMP down to 8'.

_locate in X-cut near face



Scale: NTS

LMKL

CHART: Mechanism of Loading

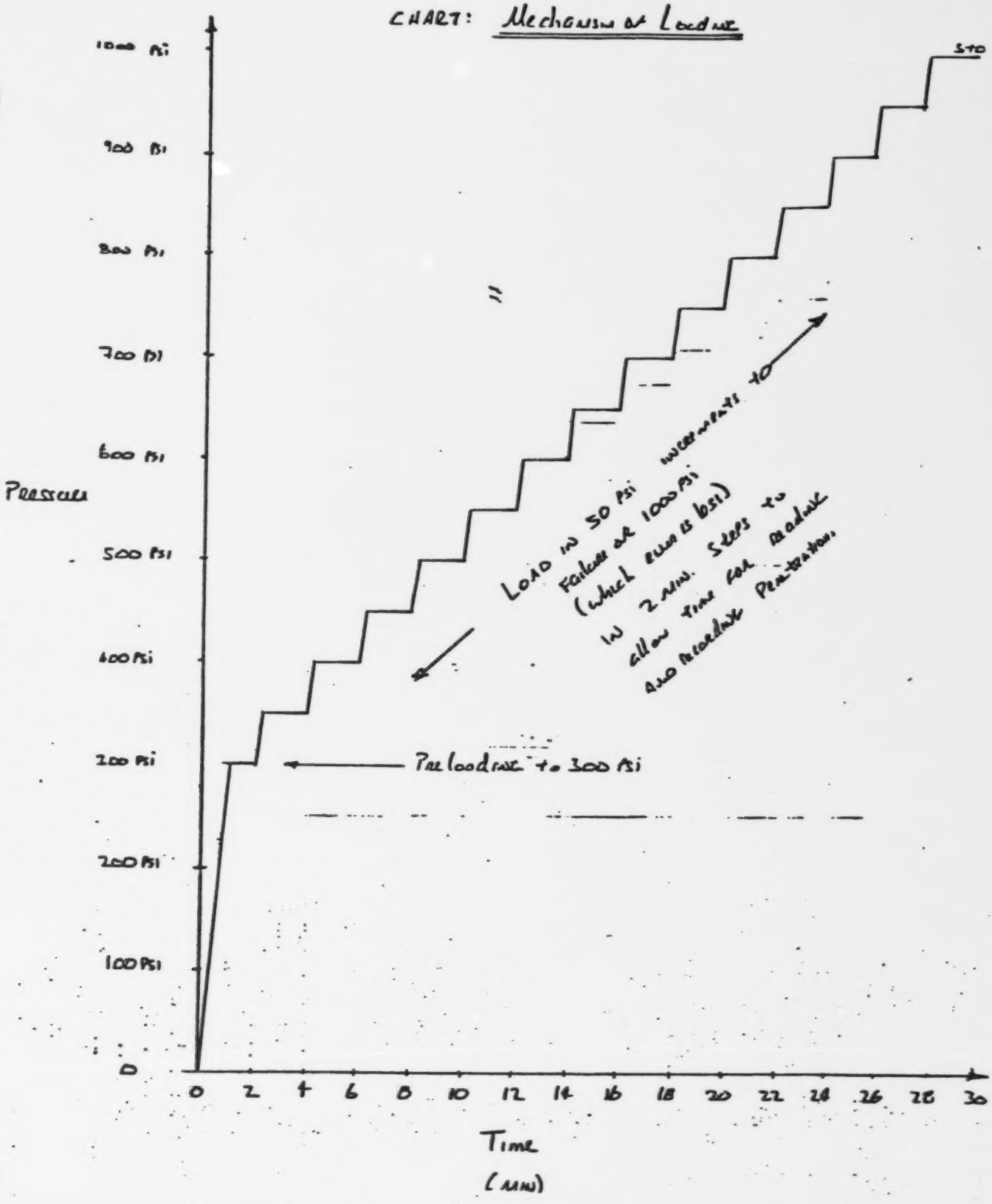


CHART: Typical Stress Strain Curve

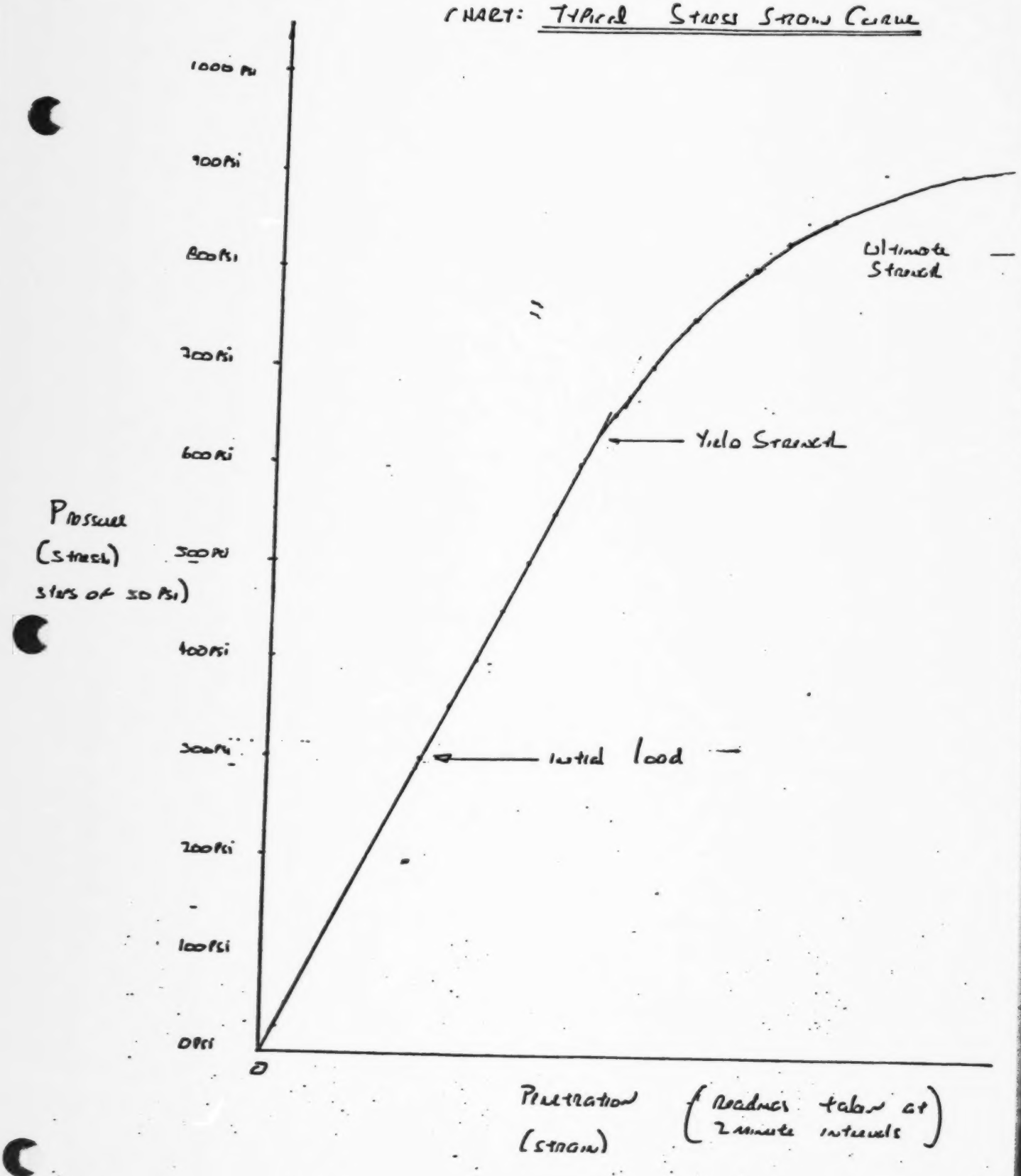


CHART- TEST 1

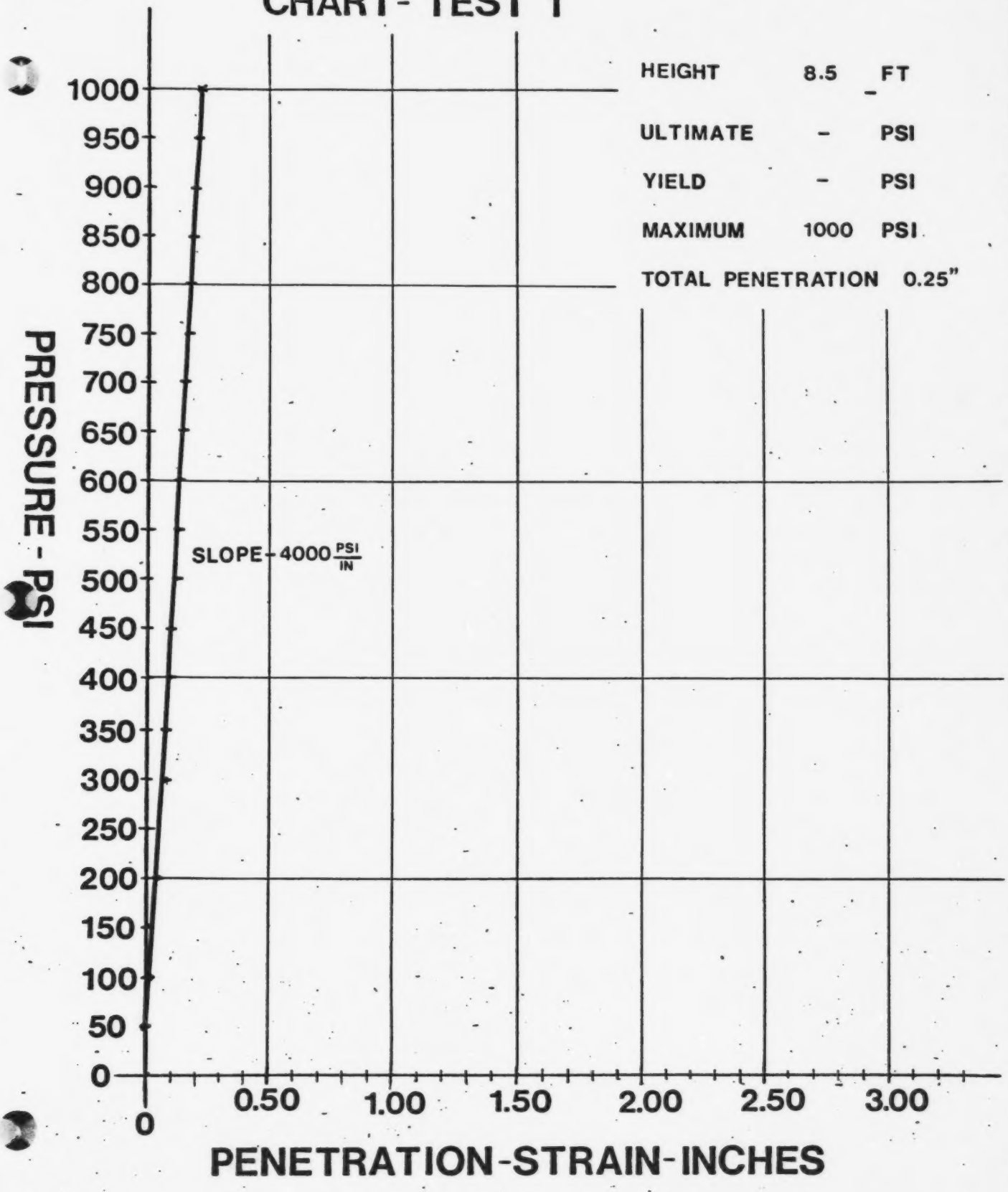
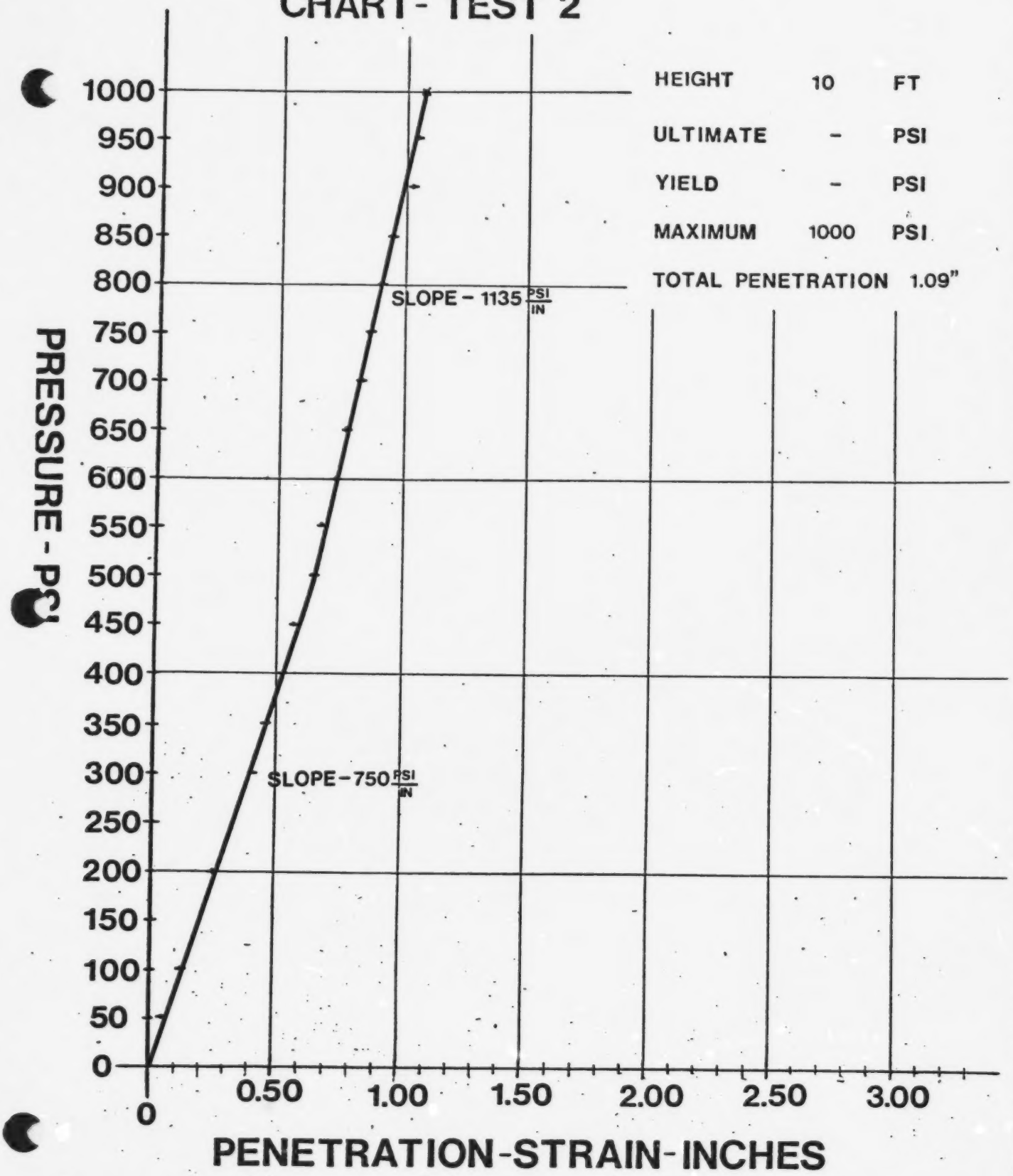

HEIGHT 8.5 FT
ULTIMATE - PSI
YIELD - PSI
MAXIMUM 1000 PSI
TOTAL PENETRATION 0.25"
SLOPE = 4000 PSI/IN
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

CHART- TEST 2


HEIGHT 10 FT
ULTIMATE - PSI
YIELD - PSI
MAXIMUM 1000 PSI
TOTAL PENETRATION 1.09"
SLOPE - 1135 PSI/IN
SLOPE - 750 PSI/IN
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

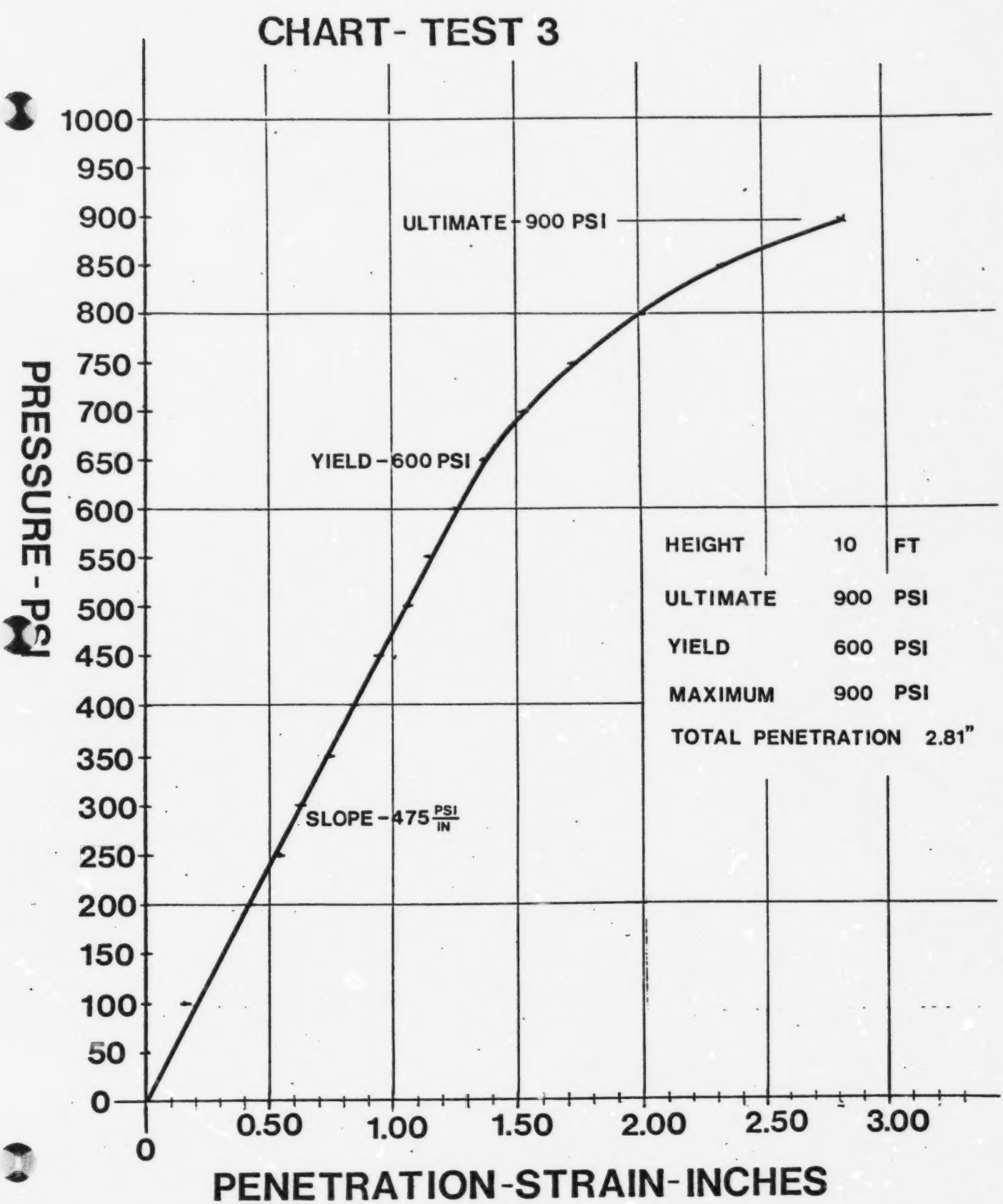


CHART- TEST 3
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
ULTIMATE - 900 PSI
YIELD - 600 PSI
SLOPE - 475 PSI/IN
HEIGHT 10 FT
ULTIMATE 900 PSI
YIELD 600 PSI
MAXIMUM 900 PSI
TOTAL PENETRATION 2.81"
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

CHART- TEST 4

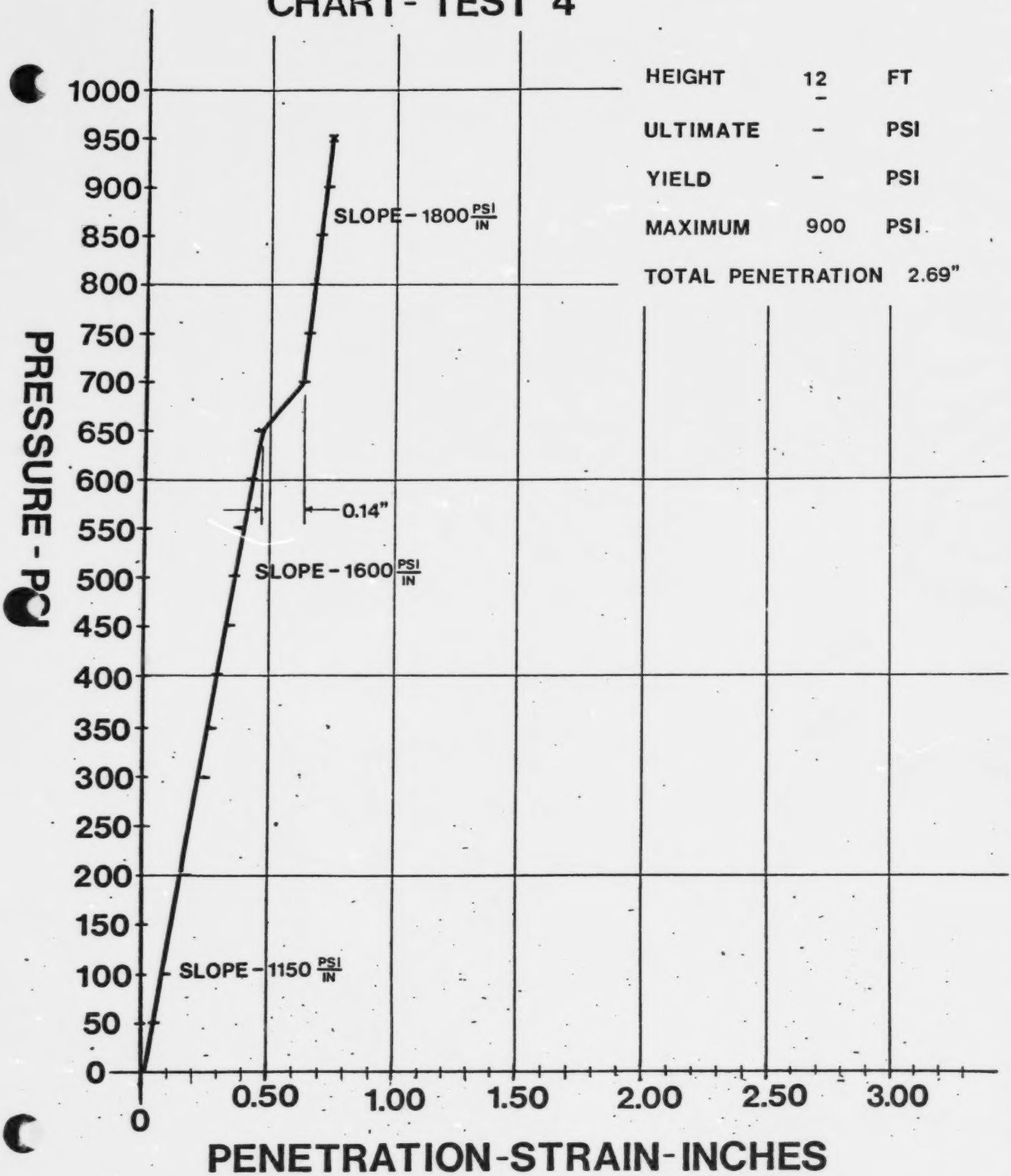

HEIGHT 12 FT
ULTIMATE – PSI
YIELD – PSI
MAXIMUM 900 PSI
TOTAL PENETRATION 2.69"
SLOPE – 1800 PSI/IN
SLOPE – 1600 PSI/IN
SLOPE – 1150 PSI/IN
0.14"
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

CHART- TEST 5

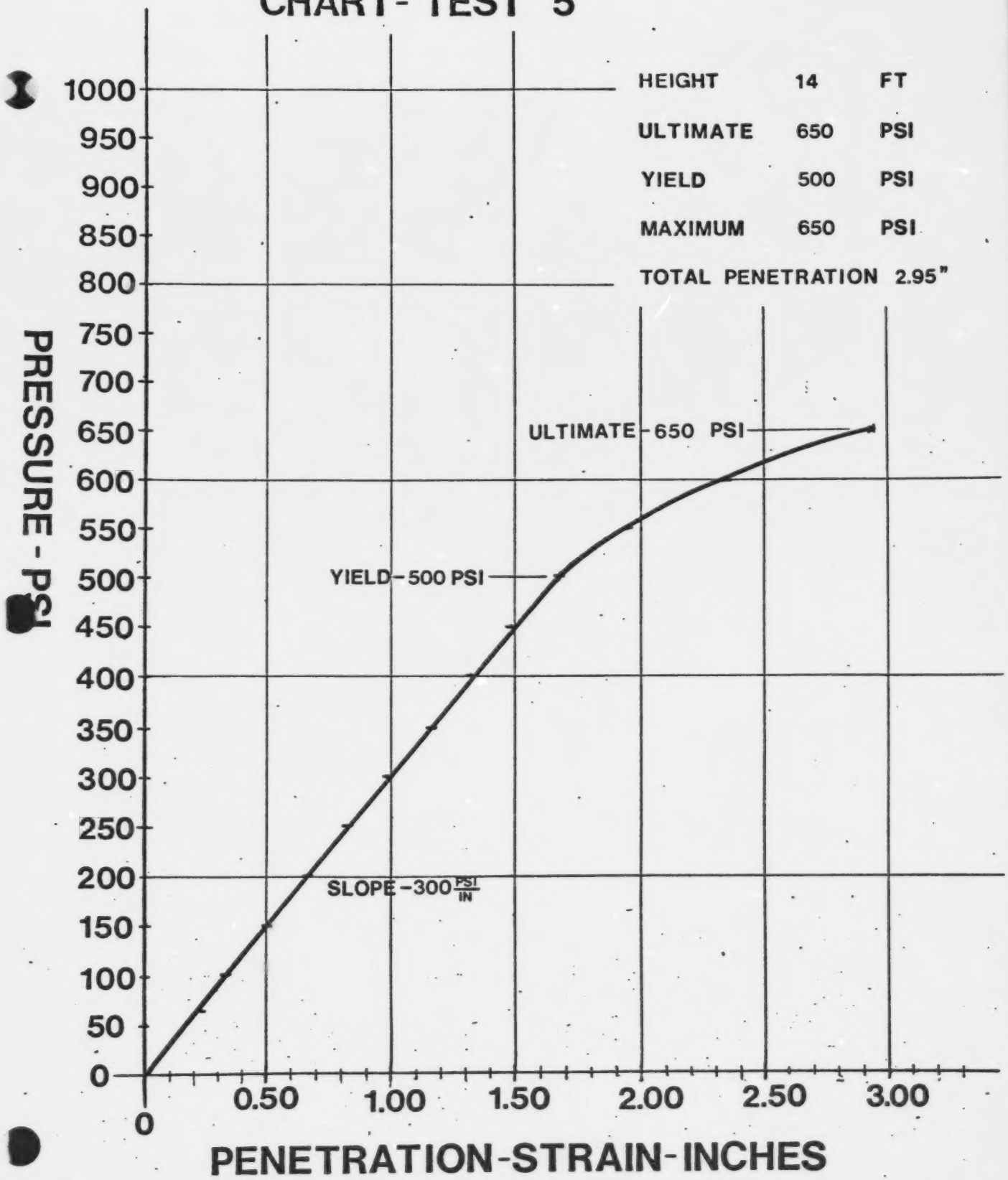

HEIGHT 14 FT
ULTIMATE 650 PSI
YIELD 500 PSI
MAXIMUM 650 PSI
TOTAL PENETRATION 2.95"
ULTIMATE - 650 PSI
YIELD - 500 PSI
SLOPE - 300 PSI/IN
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

CHART- TEST 6

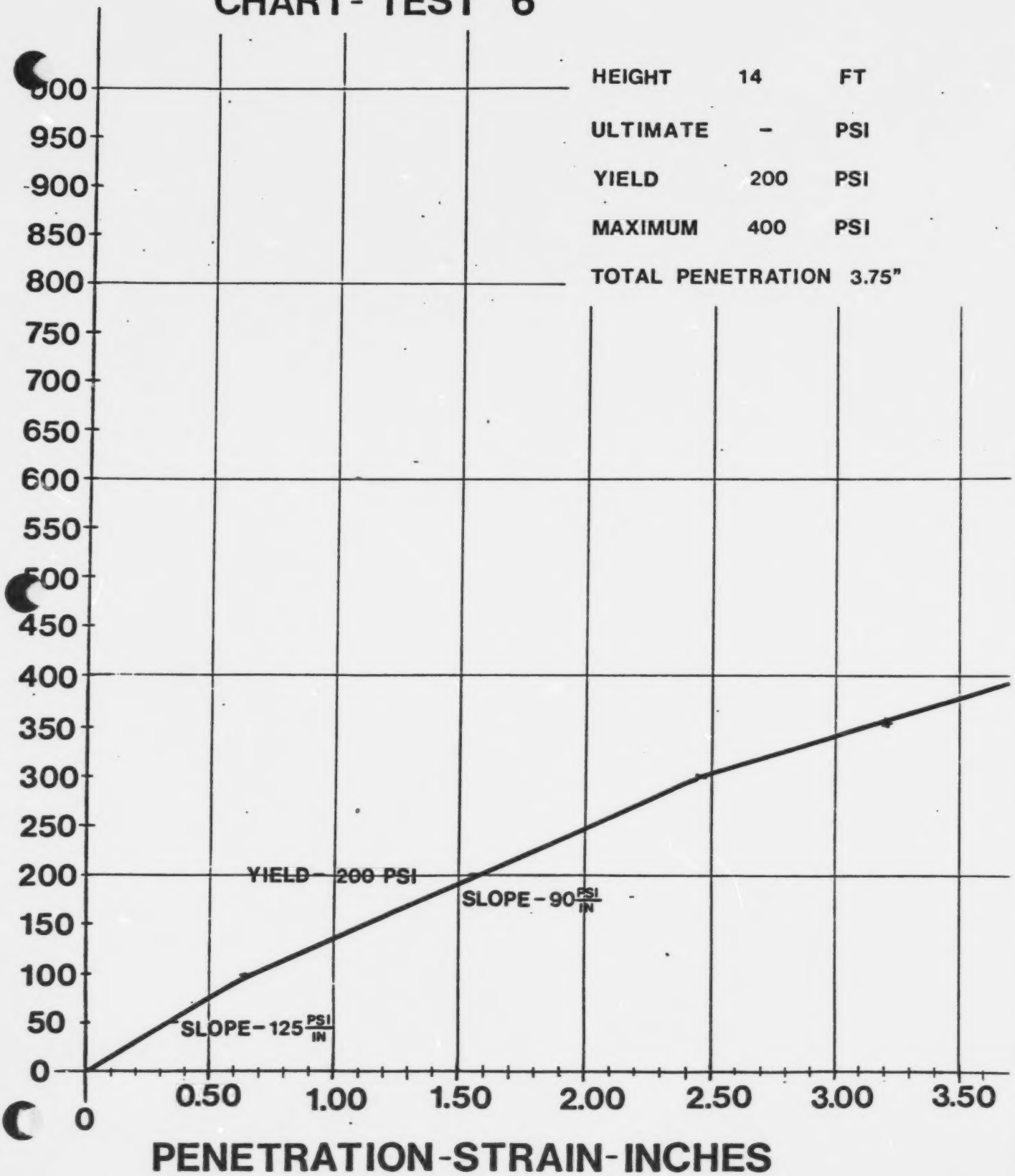

HEIGHT 14 FT
ULTIMATE – PSI
YIELD 200 PSI
MAXIMUM 400 PSI
TOTAL PENETRATION 3.75"
900
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
3.50
YIELD – 200 PSI
SLOPE – 90 PSI/IN
SLOPE – 125 PSI/IN
PENETRATION-STRAIN-INCHES

CHART- TEST 7

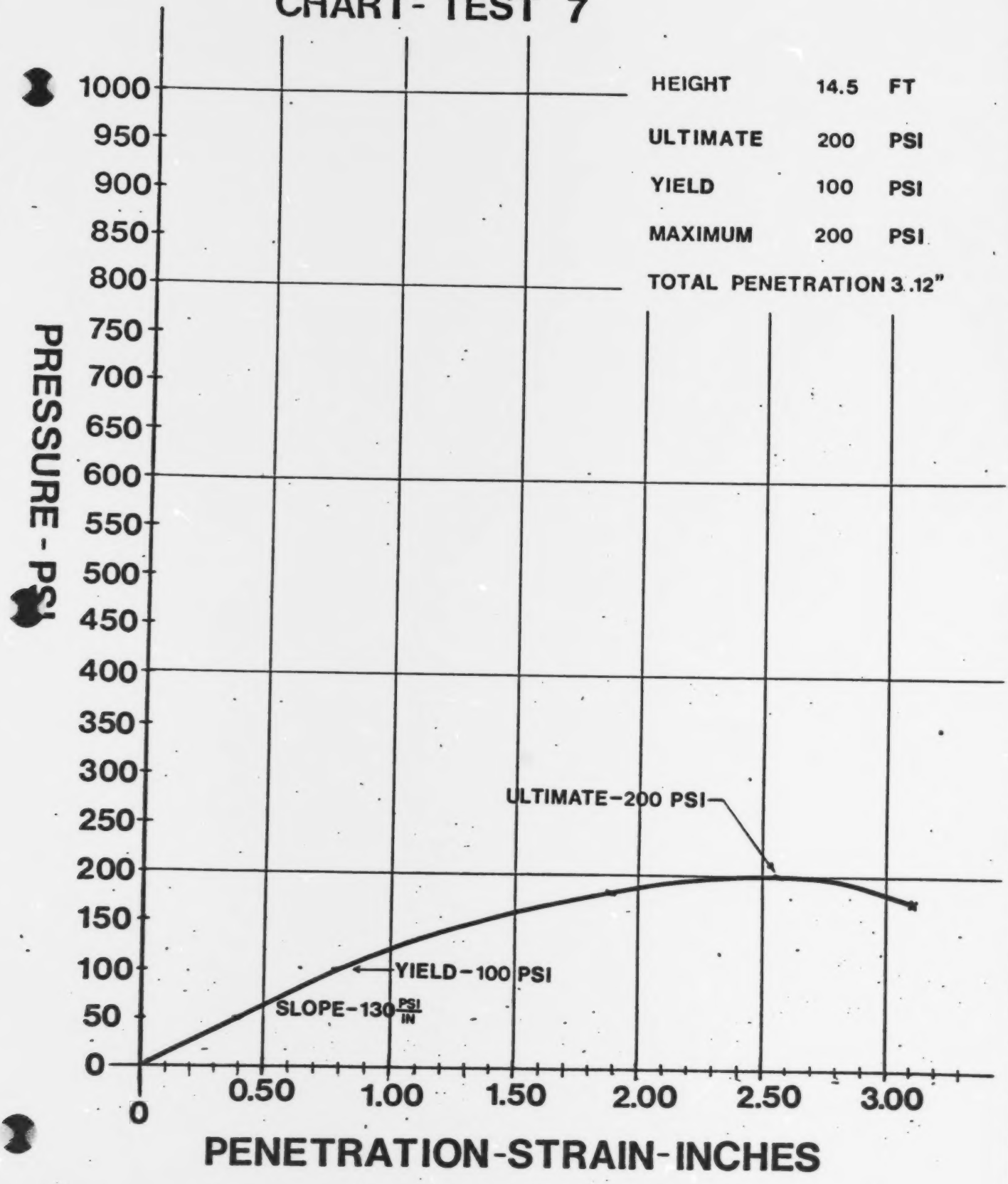

HEIGHT 14.5 FT
ULTIMATE 200 PSI
YIELD 100 PSI
MAXIMUM 200 PSI
TOTAL PENETRATION 3.12"
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
ULTIMATE-200 PSI
YIELD-100 PSI
SLOPE-130 PSI/IN
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

CHART- TEST 8

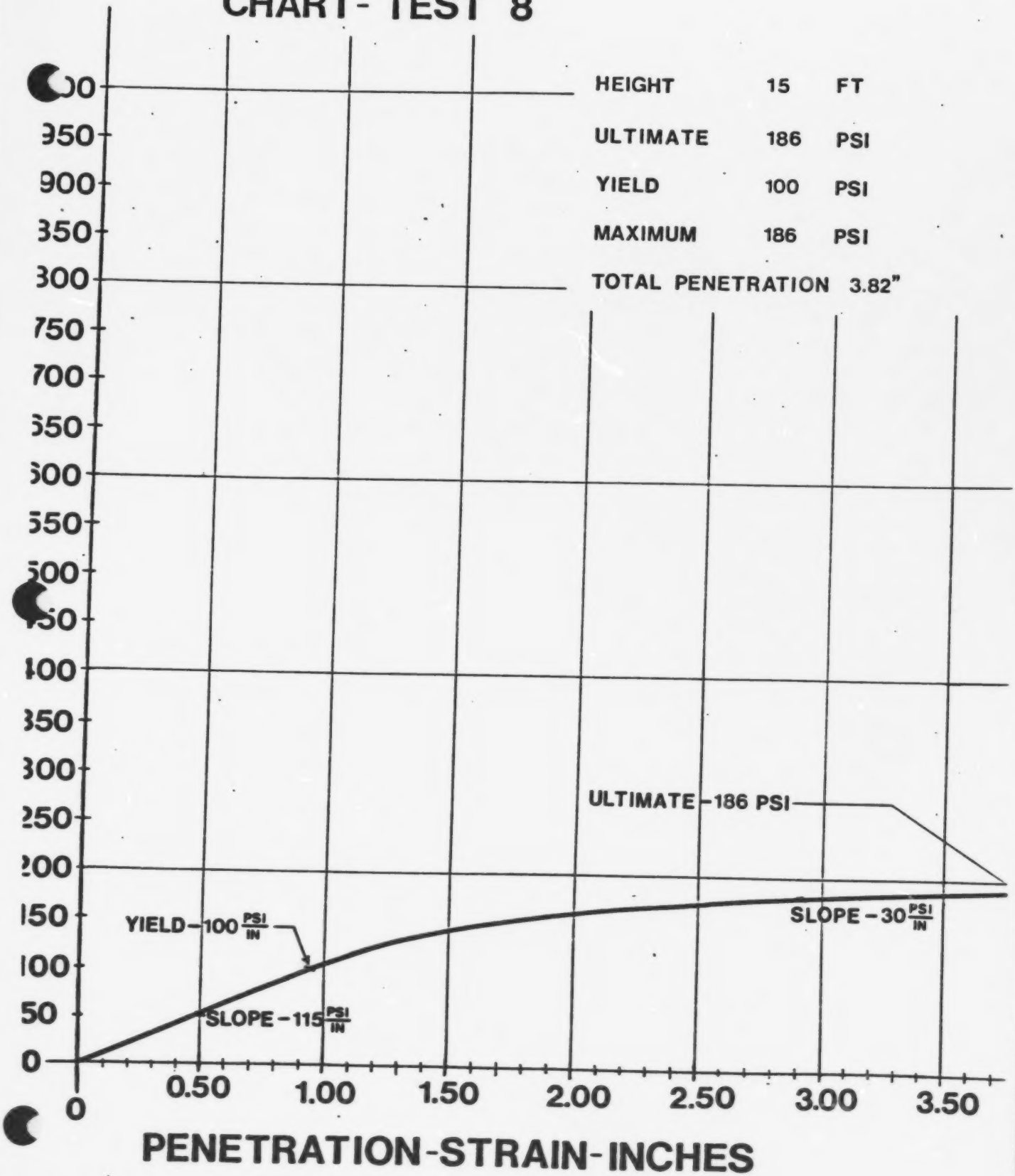

HEIGHT 15 FT
ULTIMATE 186 PSI
YIELD 100 PSI
MAXIMUM 186 PSI
TOTAL PENETRATION 3.82"
ULTIMATE – 186 PSI
YIELD – 100 PSI/IN
SLOPE – 115 PSI/IN
SLOPE – 30 PSI/IN
0
50
100
150
200
250
300
350
400
450
500
550
600
650
700
750
800
850
900
950
0
0.50
1.00
1.50
2.00
2.50
3.00
3.50
PENETRATION-STRAIN-INCHES

CHART- TEST 9

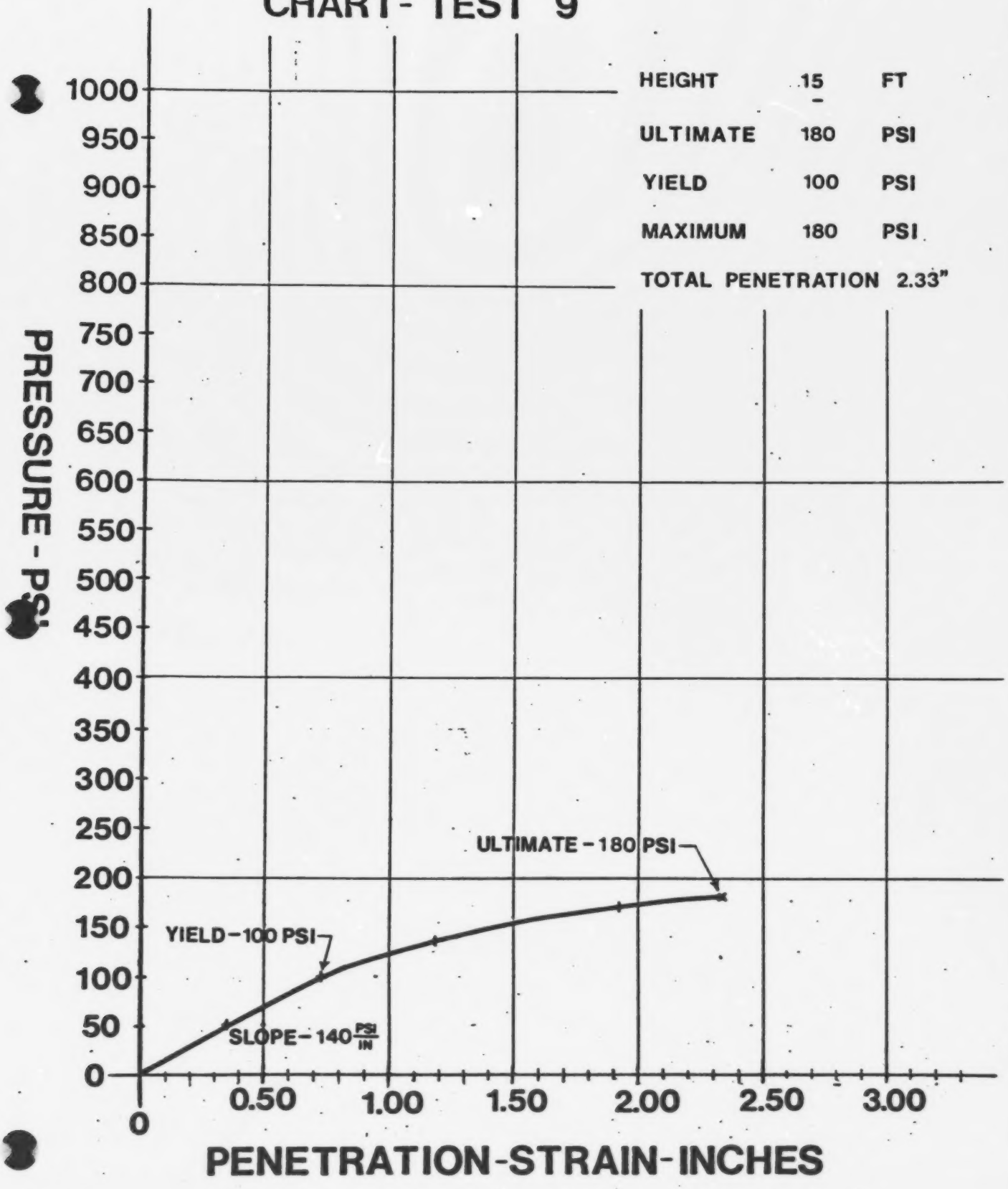

HEIGHT 15 FT
ULTIMATE 180 PSI
YIELD 100 PSI
MAXIMUM 180 PSI
TOTAL PENETRATION 2.33"
PRESSURE - PSI
1000
950
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
ULTIMATE - 180 PSI
YIELD - 100 PSI
SLOPE - 140 PSI/IN
0
0.50
1.00
1.50
2.00
2.50
3.00
PENETRATION-STRAIN-INCHES

CHART- TEST 10

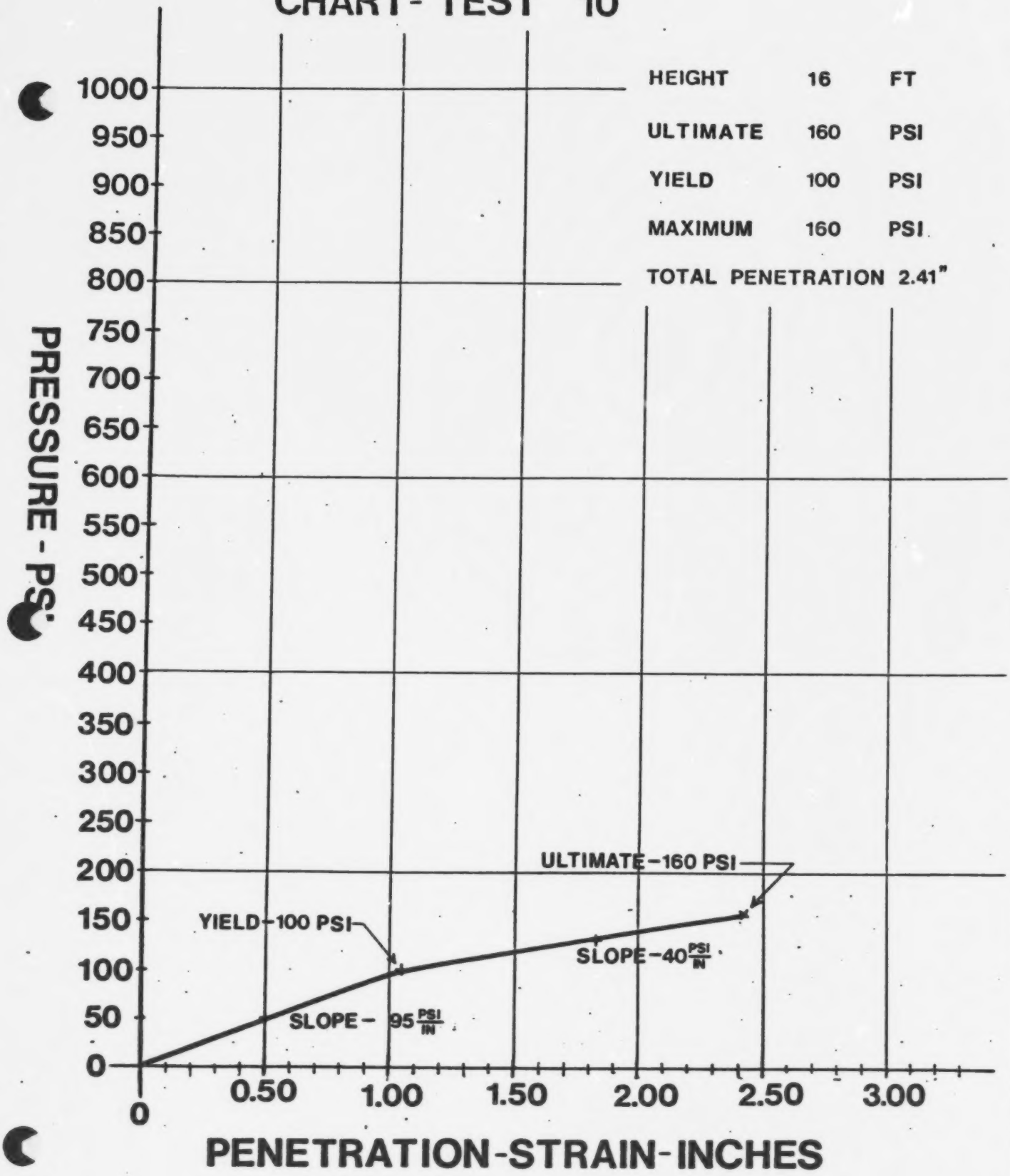


CHART- TEST 11

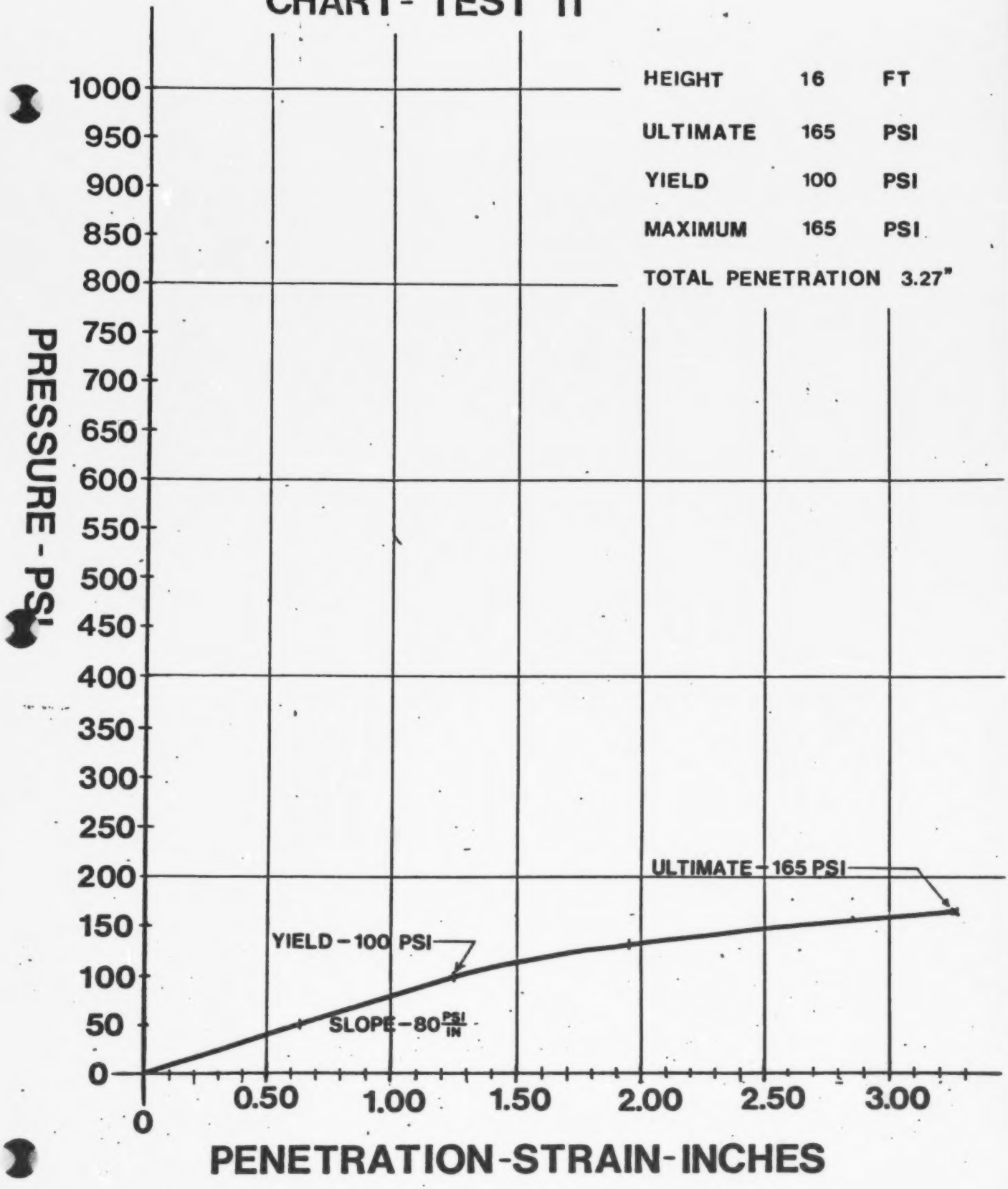


CHART- TEST 12

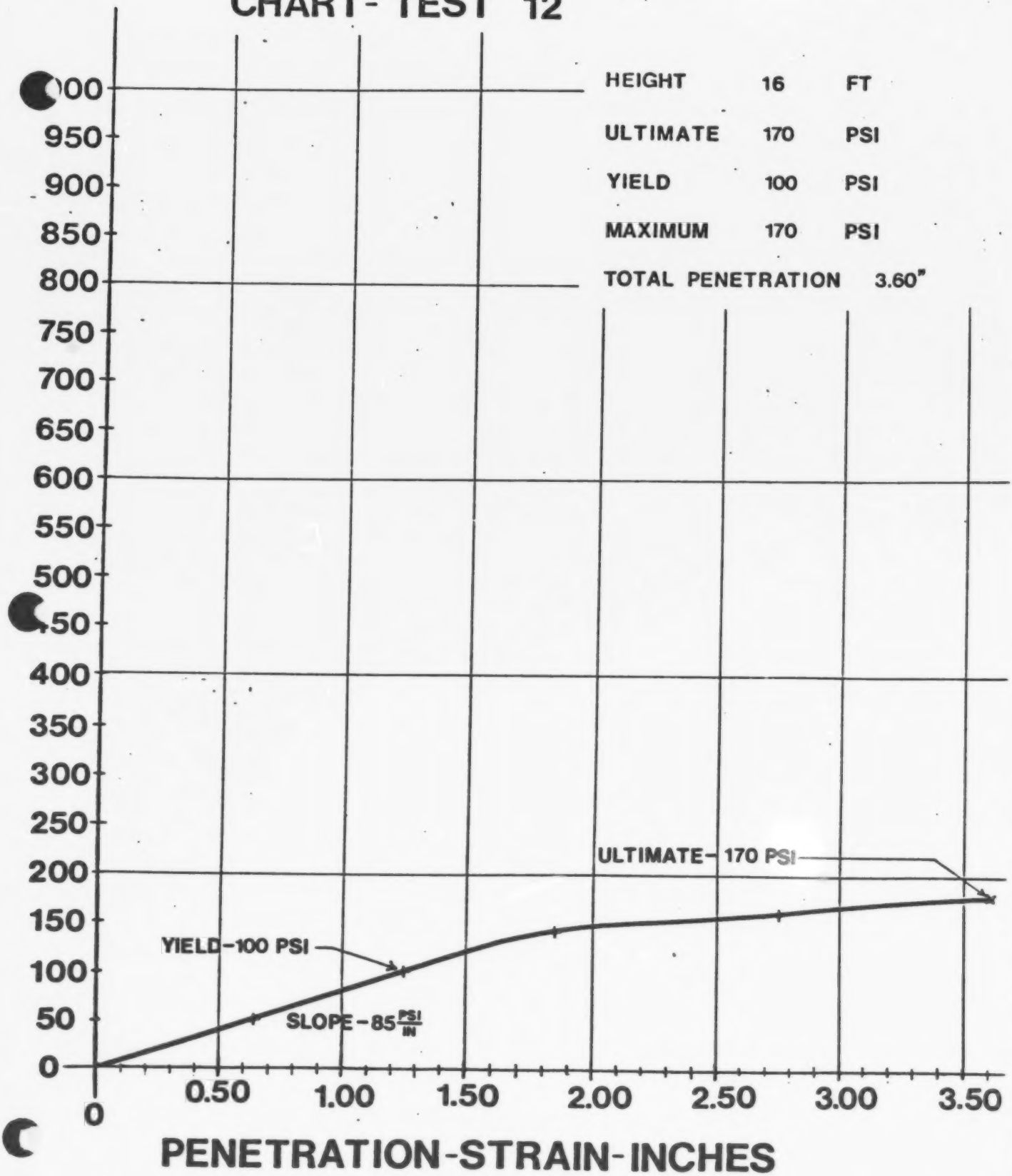
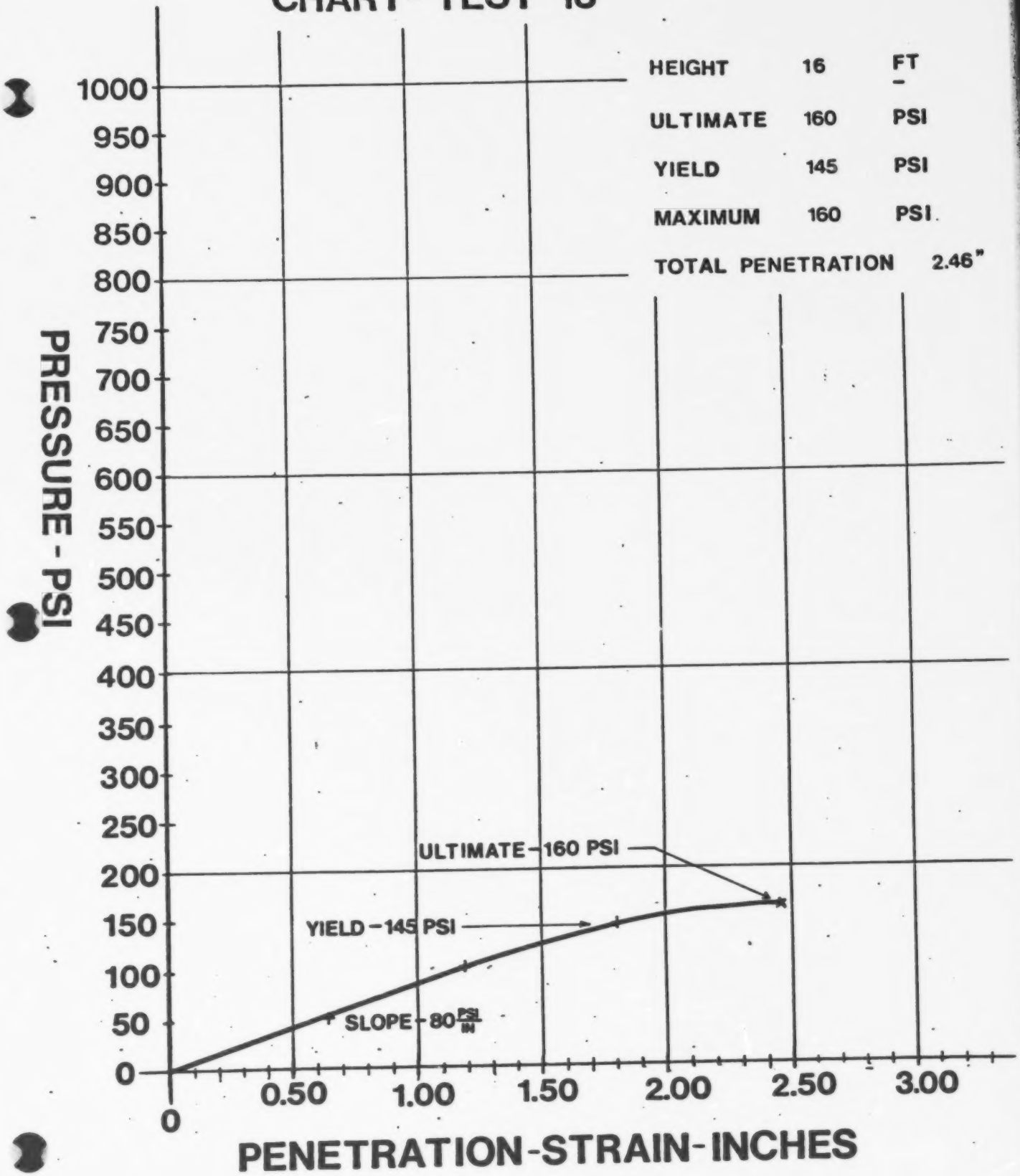


CHART- TEST 13



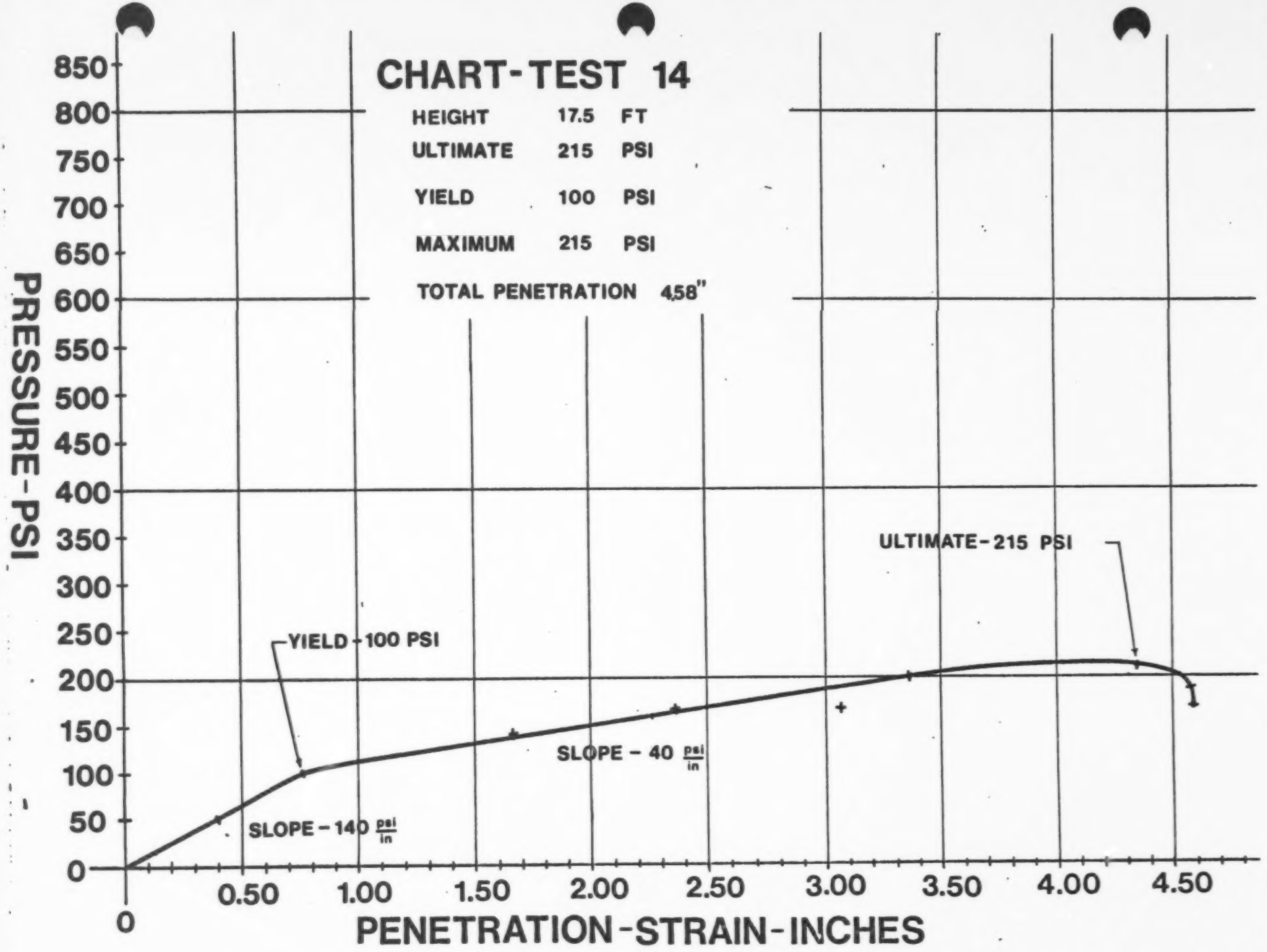


CHART-TEST 14
HEIGHT 17.5 FT
ULTIMATE 215 PSI
YIELD 100 PSI
MAXIMUM 215 PSI
TOTAL PENETRATION 4.58"
PRESSURE-PSI
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
ULTIMATE-215 PSI
YIELD-100 PSI
SLOPE - 140 psi/in
SLOPE - 40 psi/in
0
0.50
1.00
1.50
2.00
2.50
3.00
3.50
4.00
4.50
PENETRATION-STRAIN-INCHES

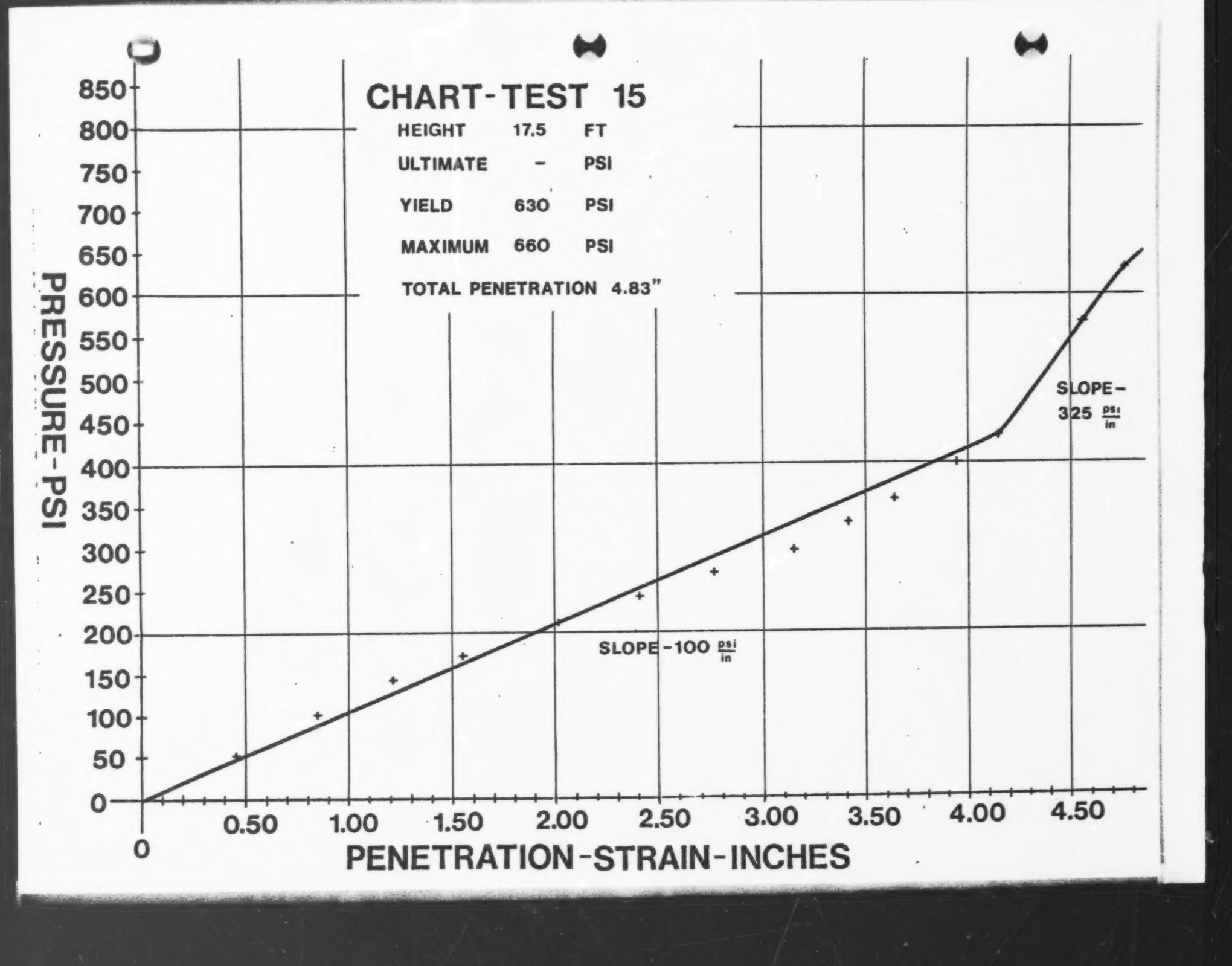


CHART-TEST 15
HEIGHT 17.5 FT
ULTIMATE - PSI
YIELD 630 PSI
MAXIMUM 660 PSI
TOTAL PENETRATION 4.83"
SLOPE-100 psi/in
SLOPE-325 psi/in
PRESSURE-PSI
PENETRATION-STRAIN-INCHES
0
50
100
150
200
250
300
350
400
450
500
550
600
650
700
750
800
850
0.50
1.00
1.50
2.00
2.50
3.00
3.50
4.00
4.50

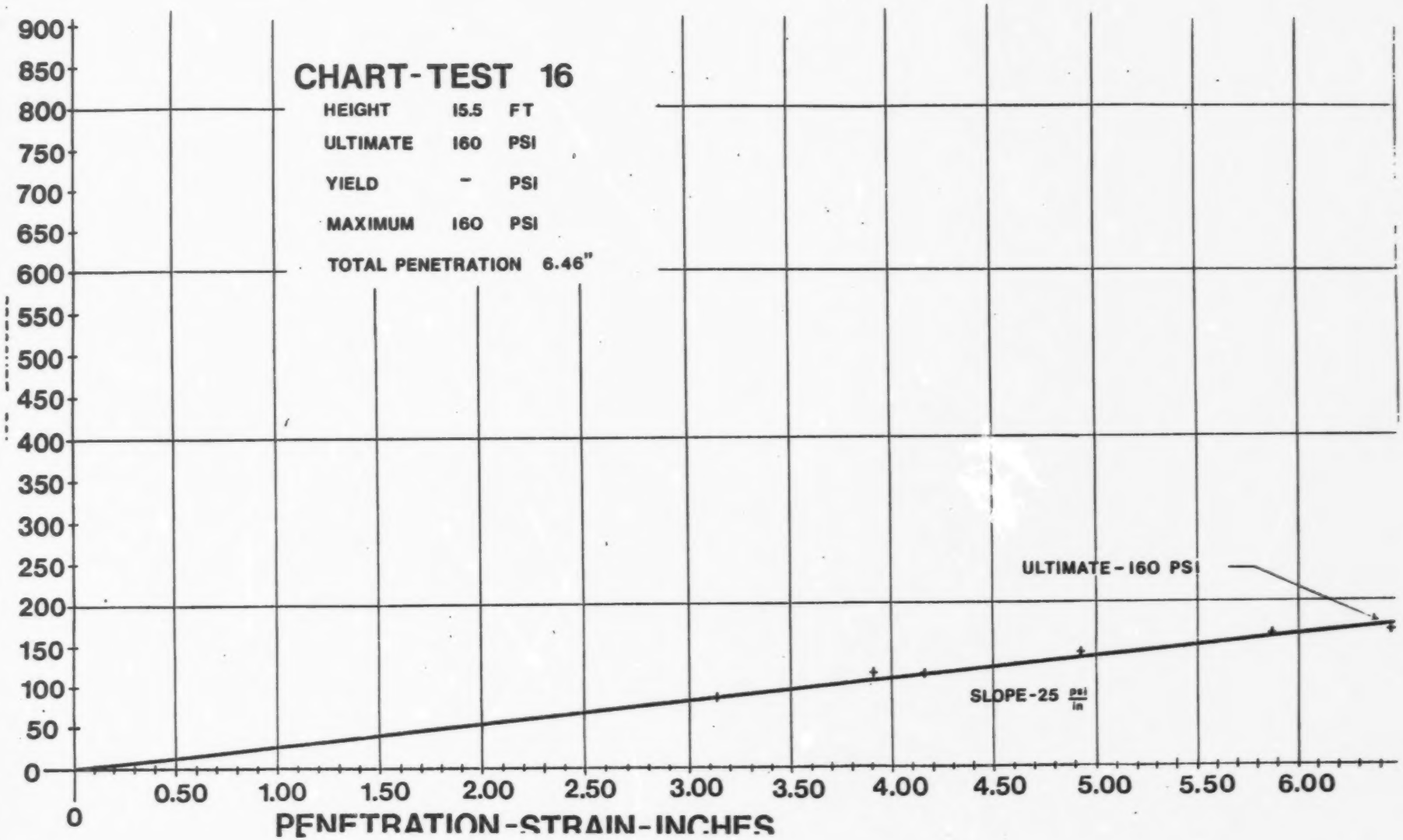


CHART-TEST 16
HEIGHT 15.5 FT
ULTIMATE 160 PSI
YIELD - PSI
MAXIMUM 160 PSI
TOTAL PENETRATION 6.46"
900
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0
0.50
1.00
1.50
2.00
2.50
3.00
3.50
4.00
4.50
5.00
5.50
6.00
ULTIMATE - 160 PSI
SLOPE - 25 psi/in
PENETRATION - STRAIN - INCHES

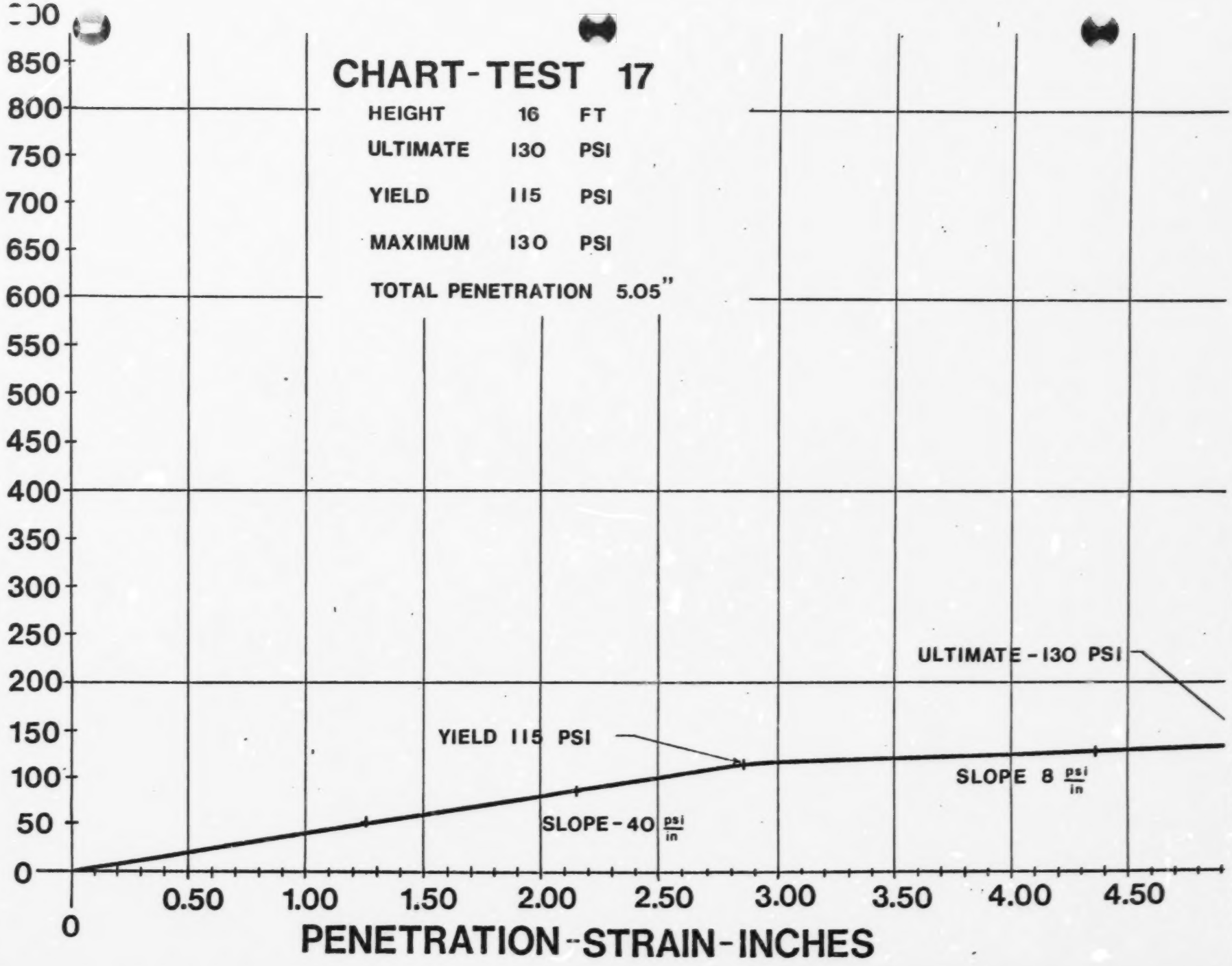


CHART-TEST 17
HEIGHT 16 FT
ULTIMATE 130 PSI
YIELD 115 PSI
MAXIMUM 130 PSI
TOTAL PENETRATION 5.05"
ULTIMATE - 130 PSI
YIELD 115 PSI
SLOPE - 40 psi/in
SLOPE 8 psi/in
850
800
750
700
650
600
550
500
450
400
350
300
250
200
150
100
50
0
0.50
1.00
1.50
2.00
2.50
3.00
3.50
4.00
4.50
PENETRATION - STRAIN - INCHES

CHART- TEST 18

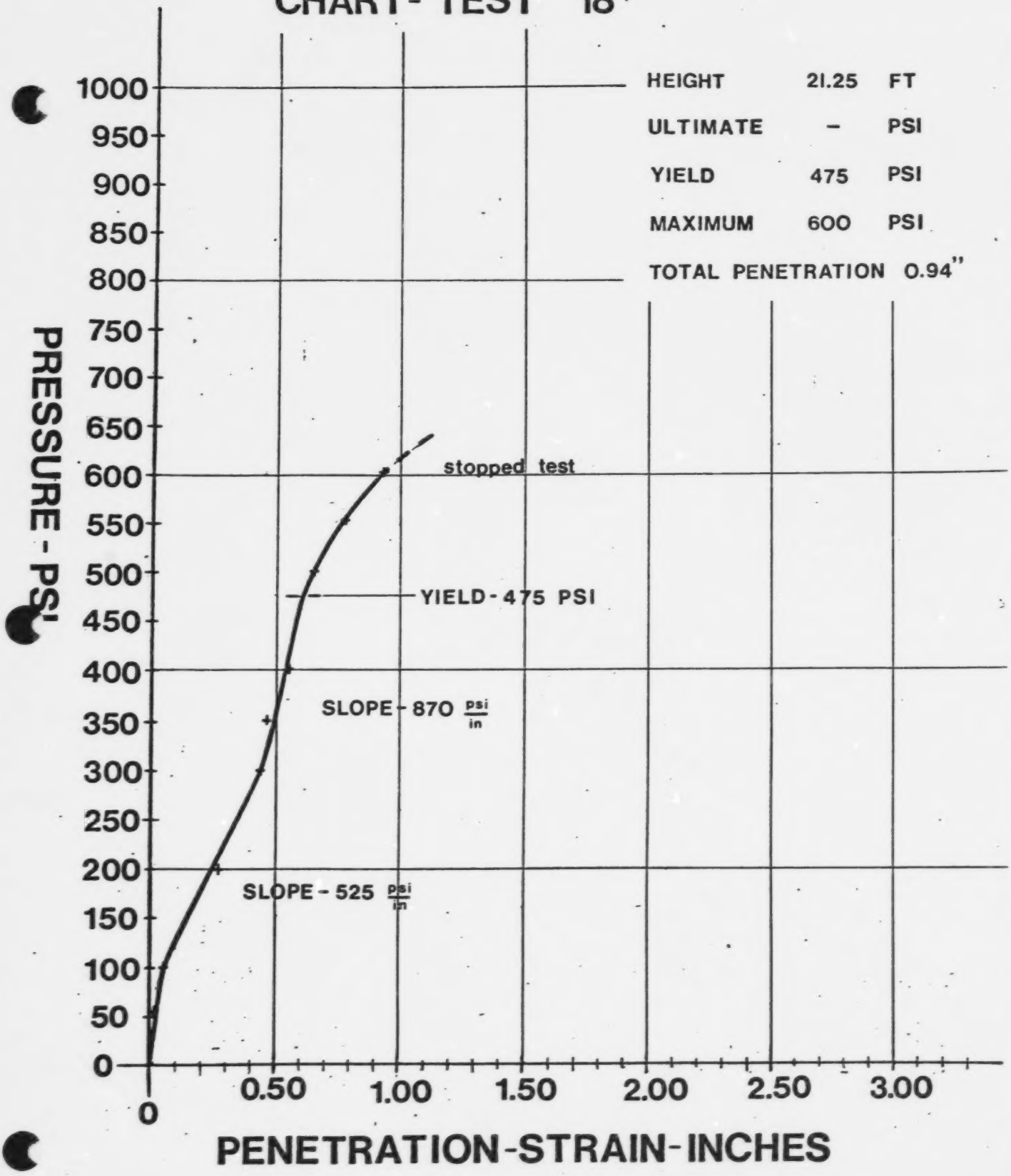
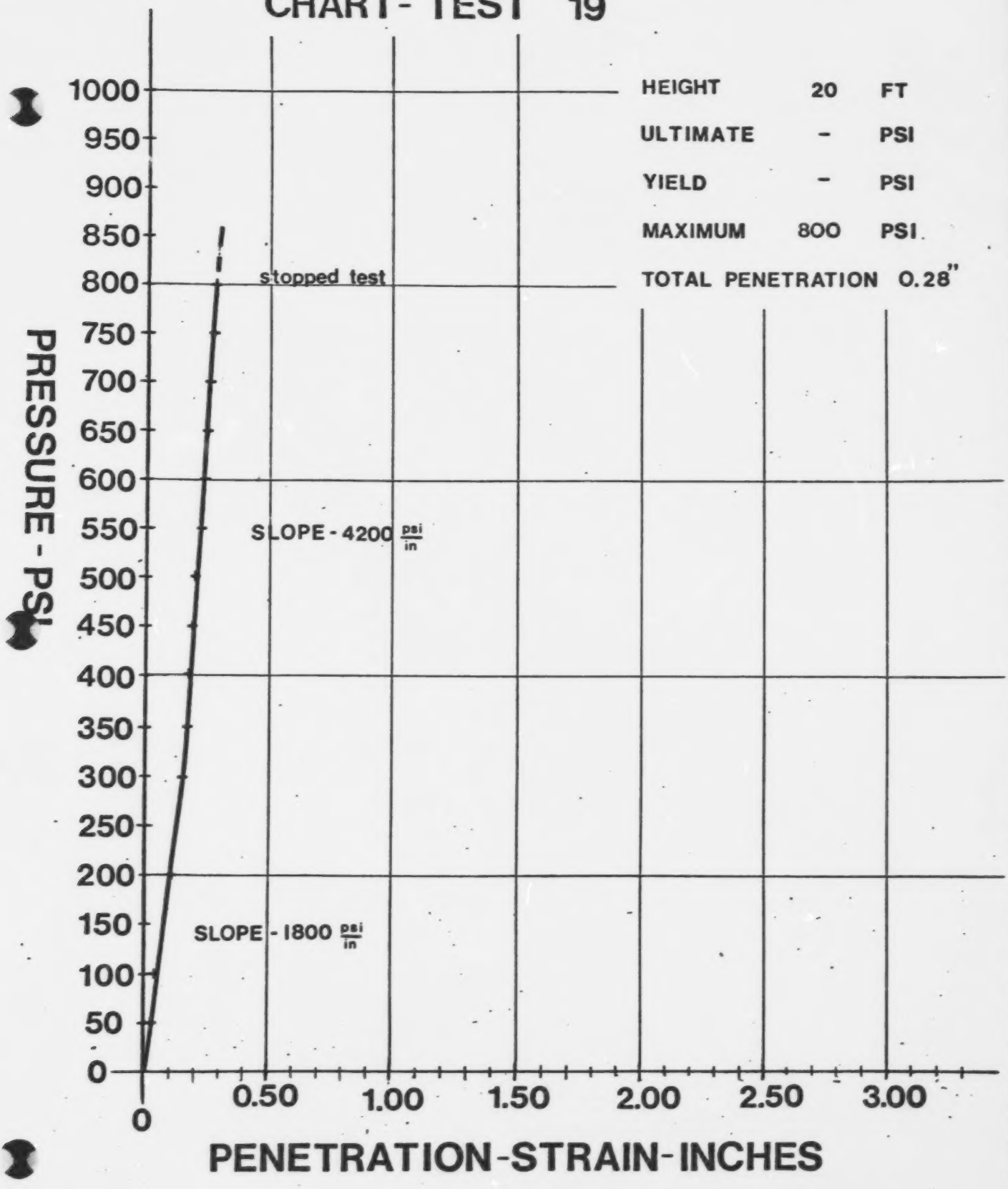


CHART- TEST 19



TEST Nº (0) Date June 13/82

Location 1" X-Cut by (1) Engineer DHG

Operator S.T

TIME	GAUGE PRESSURE	PLATE PRESSURE × 20.63/144	LOAD PSI	Deflection	DISCRIPTION
	700		100		No readings - set up equipment to test hydraulics & fitting up of equipment for 1st time
	1600		200		
	2100		301		
	2650		351		
	2800		401		
	3150		451		
	3500		502		
	3850		552		
	4200		602		
	4550		652		
	4900		702		
	5250		752		
	5600		802		
	5950		852		
	6300		903		
	6650		953		
	7000		1003		

TEST Nº ① 8'-6" DATE June 14/82

LOCATION 1st X-Cut .2N ENGINEER DHG

½ between 2N entries OPERATOR S.T.

TIME	GAUGE PRESSURE	PLATE PRESSURE × 20.63 / 144	LOAD PSI	Deflection IN.	DISCRIPTION
EVERY 2 MIN.	400			11.29	
	700		100	11.30	
	1400		200	11.34	
	2100		301	11.36	
	2450		351	11.38	
	2800		401	11.39	
	3150		451	11.40	
	3500		502	11.42	
	3850		552	11.42	
	4200		602	11.43	
	4550		652	11.44	
	4900		702	11.45	
	5250		752	11.46	
	5600		802	11.47	
	5950		853	11.48	
	6300		903	11.49	
	6650		953	11.50	
	7000		1003	11.52	

TEST Nº ② 10' Date June 14/82

Location Aprox 20' from 2W. Engineer DHG

(Distance from IN. #1 Road: 535') Operator ST.

TIME	GAUGE PRESSURE	PLATE PRESSURE x 20.62/144	LOAD PSI	Deflection IN.	DISCRIPTION
5 min between load readings	350		50	26.89	Location: water on floor ⊙ ~8' away in LN (Pond ~ 3' + 1.5' + 0.5' dp) ⊙ 30' away in 2N (10' + 2.5' + 1.5' dp)
	700		100	26.96	
	1600		200	27.10	
every 2 min	2100		301	27.25	dropping 200 PSI between load reading
	2650		351	27.30	
	2800		401	27.36	
	3150		451	27.42	
	3500		502	27.49	
	3850		552	27.52	
	4200		602	27.56	
	4550		652	27.61	
	4900		702	27.66	
	5250		752	27.70	
	5600		802	27.74	— timber cracked.
	5950		853	27.79	doubling 100 PSI between load readings
	6300		903	27.85	
	6650		953	27.89	
	7000		1003	27.93	

TEST Nº ③ 10' DATE June 15 '82

LOCATION 10 1½" water - 10' other side ENGINEER DHG

Entry from 2 (530') OPERATOR ST

TIME	GAUGE PRESSURE	PLATE PRESSURE $\times \frac{20.63}{144}$	LOAD PSI	Deflection IN	DISCRIPTION
After 1 MIN	200		100	28.16	Location - Dug 6" sluice out of wet spot to cut semi-solid base
	1600		200	28.41	- 25' from lake road Hill as for 2.
	1750		250	28.51	- water flooded site just before start of test - (before test - 1" dP from Hill sprays on nozzle.
Rubber 2 MIN	2100		301	28.62	
	2650		351	28.73	
	2800		401	28.83	
	3150		451	28.98	
	3600		502	29.05	
	3850		552	29.14	
	4200		602	29.25	
	4550		652	29.38	
	4900		702	29.52	____ after 2 min - sinks to 29.79; 4700 PSI
	5250		752	29.73	
	5600		802	30.00	
	5950		853	30.31	____ " " " " 30.34; 5400 PSI
	6100		903	30.80	____ Reached at maximum pumping rate
	6650		953	-	
	7000		1003	-	

Relaxation:

+3 min: 30.82; 5300 PSI

+3 min 30.82; 5200 PSI

+10 min 30.82; 5000 PSI

TEST Nº ④ 12' DATE June 16 '82

LOCATION 20' from ⑤, 10' from well ENGINEER DHC

(515') OPERATOR ST

TIME	GAUGE PRESSURE	PLATE PRESSURE × 2.063 / 144	LOAD PSI	DEFLECTION IN	DISCRIPTION
	250		50	22.39	Location: dry in all directions but looks as if some damp as is white rock due to spray while cutting access
	700		100	22.43	
	1600		200	22.52	
	2100		301	22.58	
	2650		351	22.61	
	2800		401	22.64	dropping 300 PSI between each reading
	3150		451	22.67	
	3600		502	22.70	
	3850		552	22.73	
	4200		602	22.76	" 600 PSI
	4550		652	22.80	closing void in floor!
	4900		702	22.97	
	5230		752	22.99	" 800 PSI
	5600		802	23.02	
	5950		853	23.04	
	6300		903	23.06	
6550	6650		953	23.08	timber on clock at back compressing too much to continue (Reach back - unload pressure on load)
	7000		1003		

TEST Nº ⑤ 14' DATE June 17/81

LOCATION 10' from wall, x-cut side ENGINEER DHC

(492') OPERATOR S.T

TIME	GAUGE PRESSURE	PLATE PRESSURE x 2.063/144	LOAD PSI	Deflection IN	DISCRIPTION
	600		57	12.61	LOCATION - 1.5' loose dust cap over site
	700		100	12.71	- dam? ground
	1600		200	13.05	- water may have run over this site from nearby sprays yesterday
RUBBY 2 min.	2100		301	13.36	- dust down 7th work
	2650		351	13.53	Shovel FELL Ft. from site
	2800		401	13.69	and cut hitting cable rod.
	3150		451	13.86	
	3600		502	14.05	
	3850		552	14.32	
	4200 ✓		602	14.70	
4100	~~4300~~		652	15.32	max Dumping Rate - stopped
	4900		702	-	
	5250		752		RELAXATION:
	5600		802		+2½ min 15.34 ; 3500 PSI
	~~5950~~		853		+10 min 15.34 ; 3100 PSI
	6300		903		+10 min 15.36 ; 3100 PSI
	6650		953		
	7000		1003		
					RESET PRESSURE FOR OVERNIGHT TEST
	3500			15.52	RELAXATION
					+1 min - 15.53 ; 3200 PSI ;
					+2 min - 15.51 ; 3180 PSI, 200 Dec
					+19 3/4 HRS 15.30 ; 2700 PSI ; 1015 Dec

TEST Nº ⑥ 14' DATE June 17/82

LOCATION ~7' from ⑤, E ENTRY ENGINEER DK

(485') OPERATOR ST

TIME	GAUGE PRESSURE	PLATE PRESSURE × 2.063 / 144	LOAD PSI	Deflection IN	DISCRIPTION
≤ 1 min	350		50	13.65	- Ground Fiber thaw ⑤, but still fractured.
	700		100	13.93	
	1400		200	14.85	
QUIET 2 min	2100		301	15.74	
	2450		351	16.40	— Sinking after Set Pressure
taking 7 min to set Pressure	2800		401	17.09	— max rate (2 Reads Previous!) 50 Less Max Reading
	3150		451		Relaxation
	3500		502		+ 1 min 17.15 ; 2300 psi
	3850		552		+ 7 min 17.16 ; 2200 psi
	4200		602		
	4550		652		
	4900		702		
	5250		752		
	5600		802		
	5950		853		
	6300		903		
	6650		953		
	7000		1003		

TEST Nº ⑦ Date June 18/82

LOCATION Center entry ; ~ 7' from ⑥ ENGINEER DHC

(480') OPERATOR ST.

TIME	GAUGE PRESSURE	PLATE PRESSURE × 20.63 / 144	LOAD PSI	Deflection IN	DISCRIPTION
	350		50	17.86	Location: duct ~8" loose and still in fractured rock
	700		100	18.25	
	1250		179	19.34	
	1400		200	①20.02	— Pumping max rate
	1200		172	②20.58	— next try could only reach 1200!
					Relaxations:
					① +25 secs 20.16; 1050 PSI, —
					② +10 min 20.59; 975 PSI; 200 m
					— — — — — — —
					+67½ hrs 20.64; 810 PSI; 8.15 cm²
					(20.98?)

PRELIMINARY PLATE LOAD TESTS - McINTYRE MINES

TEST Nº (B) 15' Date June 21/82

Location B' Fed. ⑦ Engineer DHC

(473') Operator S-T

TIME	GAUGE PRESSURE	PLATE PRESSURE $\times \frac{2063}{164}$	LOAD PSI	Deflection IN	DISCRIPTION
	350		50	28.98	
	700		100	29.42	— not yielding when stop pumping
	1250		179	① 31.40	
	1300		186	② 32.30	— max pumping rate
					Relaxations:
					① + 30 secs ; 31.41 ; 1000 psi
					② + 30 secs ; 31.31 ; 1050 psi

PRELIMINARY PLATE LOAD TESTS - McINTYRE MINES

TEST Nº ⑨ 15' Date June 21/82

LOCATION 5' FROM X-cut SIDE ENTRY. ENGINEER DHG

(975') OPERATOR ST.

TIME	GAUGE PRESSURE PSi	PLATE PRESSURE $\times \frac{2063}{144}$	LOAD PSi	Deflection IN	DISCRIPTION
	350		50	21.28	
	700		100	21.64	
	950		136	22.10	
	1200		171	22.84	
	1250		179	23.25	— max. Pump Rate
					Relaxation
					450 secs. 23.28 ; 1000 PSi

PRELIMINARY PLATE LOAD TESTS - [illegible] JOINTS

TEST Nº (10) 16' DATE June 21/82

LOCATION centre of entry ENGINEER DHG

(452') OPERATOR ST

TIME	GAUGE PRESSURE PSI	PLATE PRESSURE × 2063/177	LOAD PSI	Deflection IN	DISCRIPTION
	350		50	28.98	Location: duct 12" loose
	700		100	29.52	
	900		129	30.30	
	1100		157	30.89	— max Pumping Rate, 2 Pools

TEST Nº (11) 16' DATE June 21/82

LOCATION beside (10) (455') ENGINEER DHG

OPERATOR ST

TIME	GAUGE PRESSURE PSI	PLATE PRESSURE $\times \frac{20.63}{1.44}$	LOAD PSI	Deflection IN	DISCRIPTION
					Location: dug down 10"
	350		50	30.64	
	700		100	31.26	
	900		129	31.96	
	1150		164	33.28	- max Pumping Rate, 2 Pods
					Relaxation
					+ 30 secs ; 32.29 ; 925 PSI ; 2.40 PM
					+ 18½ Hrs ; 33.26 ; 650 PSI ; 9.00

PRELIMINARY PLATE LOAD TESTS

TEST Nº (12) 16' Date June 22/82

Location (447') Engineer DHG

Operator ST

TIME	GAUGE PRESSURE PSI	PLATE PRESSURE $\times \frac{20.63}{144}$	LOAD PSI	Deflection IN	DISCRIPTION
					Location: duc down ~8"
	350		50	31.90	
	700		100	32.50	
	1000		143	33.12	
	1100		157	34.02	
	1200		171	34.86	- max Pumping Rate 2 Pcode

TEST Nº (13) 16' DATE June 27 '82

LOCATION 5' FROM (12) : X-rat side ENGINEER DHC

(445') OPERATOR ST

TIME	GAUGE PRESSURE PSI	PLATE PRESSURE $\times \frac{20.63}{144}$	LOAD PSI	Deflection IN	DISCRIPTION
					Location: duc down 6"
	350		50	32.98	
	700		100	33.00	
	1000		143	33.64	
	1100		157	34.30	- max Pumping rate

TEST Nº (14) 17.5' DATE June 23/82

LOCATION dot on 12/ (135') ENGINEER DHG

OPERATOR ST

TIME	GAUGE PRESSURE PSI	PLATE PRESSURE × 2063/716	LOAD PSI	DEFLECTION IN	DISCRIPTION
	350		50	26.94	
	700		100	27.30	
	1000		143	28.20	
	1200		171	28.90	
	1200		171	29.60	
	1400		200	29.90	
	1500		214	30.88	
	1300		186	31.12	} attempts to go higher than 1500 PSI - no go.
	1200		171	31.12	

NOTE: total extension on lift: 10"+1/16".

TEST Nº (15) 17'-8" DATE June 23/82

LOCATION (150') ENGINEER DHG

OPERATOR ST.

TIME Min-Sec	GAUGE PRESSURE PSI	PLATE PRESSURE X 2063/144	LOAD PSI	Deflection	DISCRIPTION
					Location in soft/fractured real
0	350		50	26.40	
1.00	700		100	26.78	
1-30	1000		143	27.16	
2-10	1200		171	27.46	
3-00	1500		214	27.94	
3.35	1700		243	28.34	
4-00	1900		271	28.70	
4-40	2100		300	29.08	
5-10	2300		329	29.34	
5-40	2500		357	29.57	
6.10	2800		400	29.86	
7-00	3000		429	30.08	
7-35	3400		486	-	
8-00	4000		571	30.50	
9-00	4000		629	30.70	
9-30	4100		657	30.76	
					Relaxation
					-; 30.88; 3450 psi; 2.85 P.M
					18/4 HRS 30.86; 2375 PSI; 9.00 A.M

TEST Nº (16) 15'-6" DATE June 28/82

LOCATION (120') ENGINEER DHC

OPERATOR ST

TIME	GAUGE PRESSURE Psi	PLATE PRESSURE $\times \frac{20.63}{144}$	LOAD Psi	Deflection .N	DISCRIPTION
					Location - SOFT COAL
	350		50	4.90	
	600		86	6.02	
	800		115	6.80	
	800		115	7.06	
	1000		143	7.82	
	1100		157	8.76	
	1125		161	9.36	— MAX PUMPING RATE

TEST Nº (17) 16' DATE June 21/82

LOCATION center of entry ENGINEER DHG.

(125') OPERATOR ST.

TIME	GAUGE PRESSURE PSI	PLATE PRESSURE $\times \frac{20.63}{144}$	LOAD PSI	Deflection IN	DISCRIPTION
					Location. Soft Coal
	350		50	0.72	
	600		86	1.61	
	800		114	2.32	
	875		125	3.84	
	925		132	4.52	max Pumping Rate

TEST Nº (18) 21'-3" DATE June 20/82

LOCATION Bottom 1½' of Strong Col. ENGINEER DHC

(180') OPERATOR ST

TIME	GAUGE PRESSURE Psi	PLATE PRESSURE x 20.63/144	LOAD PSI	Deflection IN	DISCRIPTION
	400		57	18.20	
	700		100	18.23	
	1400		200	18.46	
EVERY 2 MIN	2100		301	18.61	
	2450		351	18.65	
	2800		401	18.71	
	3150		451	18.77	
	3500		502	18.84	
	3850		552	18.96	
	4200		602	19.12	— STOP: 21' high column!
	4550		652		stop to save
	4900		702		instrumentation & not
	5250		752		due to failure or
	5600		802		excessive buckling rate
	5950		853		
	6300		903		
	6650		953		
	7000		1003		

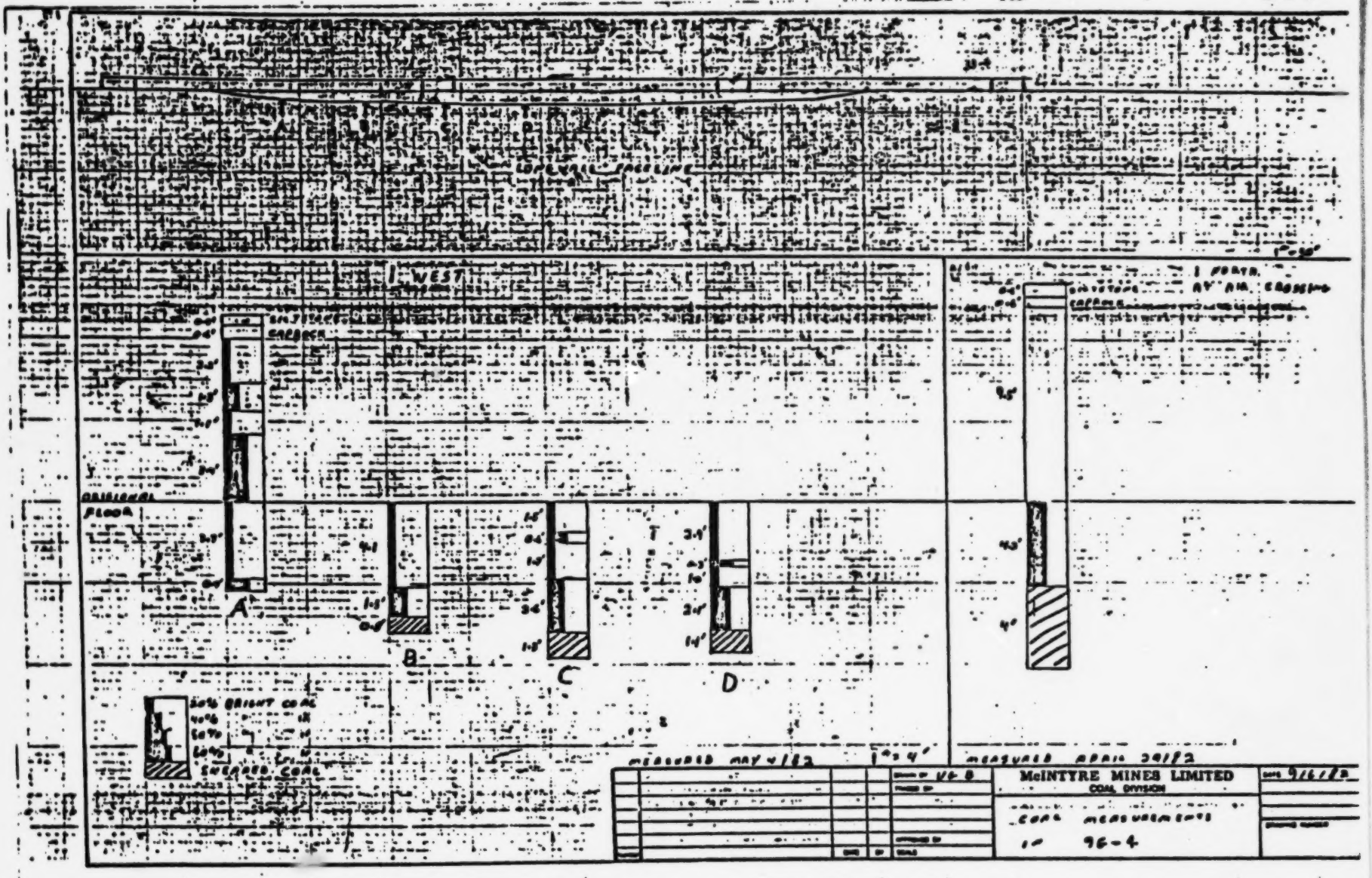
TEST Nº (19) 20 DATE June 15/82

LOCATION Centre of Entry ENGINEER DHG

Medium strength coal OPERATOR ST.

~26 to 12' above strength coal load

TIME	GAUGE PRESSURE PSI	PLATE PRESSURE x 20.63/144	LOAD PSI	Deflection IN	DISCRIPTION
	250		50	2.82	
	700		100	2.84	
every 2 MIN	1600		200	2.90	
	2100		301	2.96	
	2650		351	2.97	
	2800		401	2.98	
	3150		451	2.99	
	3600		502	3.00	
	3850		552	3.02	
	4200		602	3.03	
	4550		652	3.04	
	4900		702	3.05	
	5250		752	3.06	
	5600		802	3.00	stopped - 19' back adjusted, to protect instrumentation
	5950		853		
	6300		903		
	6650		953		
	7000		1003		Relaxation
					12 MIN ; 3.07 ; 6690 PSI
					12 MIN ; 3.07 ; 6610 PSI


1 WEST
ORIGINAL FLOOR
A
B
C
D
1 NORTH
MEASURED MAY 4/82
MEASURED APRIL 30/82
McINTYRE MINES LIMITED
COAL DIVISION
CORE MEASUREMENTS
96-4

COAL MINING RESEARCH CENTRE

Supplement to Preliminary Plate Load Bearing tests, McIntyre Mines, Grande Cache.

by D.H. Green

CMRC Report #82/49-C JULY 1982

ADDENDUM TO REPORT #82/46-C

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2. Plate Size Determination 12" x 12" 1

COROLLARIES:

A. Insitu Tests of Bearing Capacity of roof and floor in selected bituminous coal mines. USBM R1-7406.

B. Pictures of Plate Load Test Equipment.

Supplement to Preliminary Plate Load Bearing Tests, McIntyre Mines.

1. Description of Test Equipment

The Plate Load test equipment consists of

(a) Enerpac 100 ton capacity hydraulic cylinder RR 10013

(b) Enerpac double acting pump - P464

(c) 8' nominal dia., ¼" wall, 50,000 lb. yield (CSA G40. 21-50W) structural steel column in sections of 6', 4', 3', 2', 1'. Each section bolts together by means of flanges to form a unit operable at heights from 3' to 21'. Design capacity of column is 100 tons at 20'. Unit is complete with stabilizers to hold unit in place and a 3' long rod marked off in 0.01". Unit comes with a 12" x 12" square 1" thick steel bottom plate; 12" x 18" top plate wooden blocking is used to increase roof bearing area to approx. 10 ft². Alignment of sections to ±0.01", faces of column, machined right angles to length to match flanges complete with alignment pins on face of flanges.

Miscellaneous - 10,000 psi hydraulic hoses, fittings, 6" pressure guage to ½%, steel storage box, operating manuals, 3/4" unc bolts and nuts (CSA A325 or better), welding rod used - 7018 (70 ksi) spec. - CSA W591 All components are steel.

NOTE - Corollary B pictures of Plate Load Test Equipment.

2. Plate Size - Determination 12" x 12"

The USBM ran numerous tests using various sizes of plates in hard coal in longwall mines. They found that the average bearing capacity of a coal seam tends to become constant when tests are made with rectangular (but not circular) plates of large area. They ran a series of 80 tests in 9 mines using plates of 6" x 6" to 12" x 12" and determined that the bearing capacity does not vary significantly with the size of the bearing plate within this range. They also state that the results of plate load tests may be used without any need for extrapolation. See Corollary A for copy reference: In Situ tests at Bearing Capacity of Roof and Floor on selected Bituminous Coal Mines USBM RI-7406.

Numerous other tests have been done in the U.S. and the NCB in the U.K. with 12" x 12" plates.

These tests indicate that a 12" x 12" plate gives a very good approximation of the loading characteristics of the bases on the floor for longwall supports. These support bases commonly are some 20" wide.

Enclosed are two examples of plate load tests done previously in western Canadian coals, reference: Couetdic J.M. and Barron K., Plate Load Testing as a method of assessing In situ strength of Properties of western Canadian Coal, Int. J. Rock. Min. Sci. and Geom. Abs. Vol. 12, 1975.

Note: Both USBM and Dames and Moore have used 100 ton plate load equipment.

15.360

COAL MINING RESEARCH CENTRE

IN SITU TESTS OF BEARING CAPACITY OF ROOF AND FLOOR IN SELECTED BITUMINOUS COAL MINES

A Progress Report - Longwall Mining

By A. J. Barry and Oscar B. Nair

report of investigations 7406

UNITED STATES DEPARTMENT OF THE INTERIOR

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IN SITU TESTS OF BEARING CAPACITY OF ROOF AND FLOOR IN SELECTED BITUMINOUS COAL MINES

A Progress Report—Longwall Mining

by

A. J. Barry[1] and Oscar B. Nair[2]

ABSTRACT

A practical method has been developed and field tested which provides an estimate of the bearing capacity of mine roof and floor. The relationship between penetration of the strata and imposed loads is determined by in situ tests. Information of this type is needed to determine the maximum allowable pressure that may be applied by self-advancing supports on roof and floor of a longwall face without causing a punching type of failure.

INTRODUCTION

This study is part of a continuing effort by the Ground Control Technology Group of the Bureau of Mines to develop engineering information and to determine parameters that will be of use in the design of longwall mining systems.

With the advent of self-advancing hydraulic supports or chocks having design capacities of up to 600 tons, and the prospect that longwall mining will find increasing popularity in U.S. coal mines, there is a great need for engineering information that will help the operator to select his longwall support equipment and minimize the possibility of failure after installation underground.

Chocks of large yield loads currently being used in the United States can and do periodically apply large bearing pressures to the roof and floor strata at operating longwall faces. High bearing pressures can be reduced either by increasing the contact areas of the bearing plates or by reducing the yield load of the chocks below the design value. If the mine roof and/or floor strata are weak in bearing strength, large bearing pressures can cause the chocks to penetrate into the strata and cause operational difficulties. These difficulties can be minimized by determining the bearing strength of the mine strata prior to installation of the support equipment. Support equipment considered safe and efficient for the conditions existing in the particular

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[2]Mining engineer.
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mine may then be selected such that the average bearing pressures exerted by the support will be less than the bearing strength of the strata.

The objective of this investigation was therefore (1) to develop a convenient, diagnostic, bearing capacity test and (2) to conduct sufficient tests in order to determine its effectiveness and reliability.

ACKNOWLEDGMENTS

J. S. Miller, formerly mining engineer, Bureau of Mines, assisted with the field testing. The assistance and cooperation received from officials and miners at the different mines is gratefully acknowledged.

STUDIES BY OTHER INVESTIGATORS

Various types of apparatus have been used to determine the inplace strength of coal measures and associated roof and floor strata. Principal among them are penetrometer tests (2, 5-6, 10)[3] and plate bearing tests (2-3, 6-8, 10). In addition, the civil engineering literature abounds with descriptions of methods to determine the bearing capacity of floors and foundations based on either a Soil Mechanics or a Theory of Elasticity interpretation (1, 9, 11-15).

With the usual penetrometer type of apparatus, a round rod of relatively small cross-sectional area, having either a conical, hemispherical, or flat end, is forced into the stratum being tested. The maximum resistance developed for a predetermined amount of penetration is measured. The equipment is usually very simple, portable, and designed for performing numerous tests over different areas with little difficulty. Test results show that, in general, the resistance of coal to penetration is as much as five times the uniaxial crushing strength of coal (3).

Later investigators (2-3, 6-8, 10) used a combination of the penetrometer and plate-load type of apparatus. Circular plates ranging in area from 0.014 in.^{2} (0.09 cm^2) to 108.5 in.^{2} (700 cm^2) were loaded until the stratum on which the plate was acting failed. Jenkins (7-8) plotted the rupture load (load at failure), R, against the plate diameter, D, and found a relationship $R = KD^n$ where K and n are, respectively, constants governed by the strength of the floor and the nature of loading between the bearing plate and the stratum tested. Jahns (6) plotted the specific breaking strength (load at failure divided by bearing plate area) against bearing plate area on semilog paper and produced a series of curves for different types of rocks (Rock Strength Standards). Both investigators then extrapolated the curves to predict the bearing capacity of rock when loaded by mine props.

Coates and Gyenge (3) employed plate loading to determine the Modulus of Deformation and the Bearing Capacity of the rock tested. Rigid, circular bearing plates, 4 to 8 in. in diameter (12.57 in.^{2} to 50.26 in.^{2} area), were

[3]Underlined numbers in parentheses refer to items in the list of references at the end of this report.

used to test three types of rock. Results were not conclusive regarding the bearing capacity of the rock types, although a correlation was found between the modulus of deformation and the bearing capacity.

Dulaney (4) used rigid square bearing plates, ranging from 1½ to 12 inches square (2.25 in.^{2} to 144 in.^{2} area), to determine the bearing strength of mine floors in six different coal seams. The plates were placed on a ½-in.-thick layer of plaster of paris laid at each test location to minimize stress concentrations and to insure uniform loading.

None of the above-mentioned investigators extrapolated their results to determine the maximum allowable bearing pressure applied by footplates and canopies of longwall self-advancing supports.

The bearing pressure applied by the footplates and canopies of existing longwall self-advancing supports does not exceed 1,000 psi at the designed yield load value. The size of the footplates and canopies ranges from 12 by 12 in. to 36 by 108 in. and possibly larger.

From an engineering standpoint, the most desirable plate loading test is one that requires a minimum of extrapolation to the full size application. Consequently, the authors have used the largest possible plate sizes within the bounds of practicality in order to keep the extrapolation, if any, to a minimum.

DEVELOPMENT OF APPARATUS

Initially, the apparatus consisted of a 30-ton hydraulic prop, a 5,000-psi pressure gage and recorder, a 10,000-psi hand-operated hydraulic pump, dial gages to measure roof and floor penetration, prop extensions, and circular penetrometer plates of 1-, 2-, 3-, 4-, 5-, and 10-inch diameters.

Preliminary tests were made at two locations in a Colorado coal mine (mine 1). At the test locations, the floor consisted of weak-bedded shales and the roof consisted of at least 3 ft of coal overlain by shales. Most of the tests were made in the vicinity of an area being considered by mine management for longwall mining.

Table 1 gives the results of the tests. An attempt was made to keep the rate of loading constant during testing. The bearing pressure at which the strata initially failed was determined, as well as that for the failure, if any, of successive strata. Each test was concluded when a penetration of the plate into the strata amounted to 4 in. (a penetration of a support unit into the strata of greater than 4 in. would be undesirable from ground control considerations) or when the load capacity of the prop was reached (30 tons). Throughout this report the expression "bearing pressure" implies the average pressure over the area of the bearing plate, and is calculated from

$$\text{Bearing pressure} = \frac{\text{Applied load}}{\text{Area of bearing plate}}. \qquad (1)$$

TABLE 1. - Bearing capacity tests at mine 1, Colorado, circular plates

(Capacity of system = 30 tons)

Bearing plate diameter, in.	Bearing pressure, psi		
	At initial failure	At 2d failure	Maximum reached during test
TESTS ON FLOOR AT LOCATION A			
1	6,080	8,240	8,240
2	5,600	7,800	7,800
3	3,900	3,820	3,900
4	No failure	-	3,000
TESTS ON ROOF AT LOCATION B			
1	6,400	9,600	9,600
1	4,500	9,600	9,600
1	6,400	8,000	16,000
2	5,600	6,800	6,800
2	2,000	9,600	10,400
2	4,000	6,800	6,800
3	6,750	5,150	6,750
3	6,050	-	6,050
4	3,350	5,000	5,000
4	No failure	-	5,000
TESTS ON FLOOR AT LOCATION B			
1	1,300	-	3,200
1	1,600	-	2,840
1	4,050	-	4,320
2	1,600	-	1,600
2	2,400	-	2,400
2	4,200	-	4,200
3	2,130	-	2,130
4	1,240	-	1,240
4	800	1,500	1,500
4	700	-	700
5	830	-	830
5	1,090	-	1,090
5	1,130	-	1,130
10	No failure	-	800
10	640	-	640
10	750	-	750

Analysis of the results indicated that the apparatus had to be modified to accommodate larger penetrometer plates before realistic extrapolation of data to chock-size bearing areas could be done. The apparatus was accordingly modified and further tests were conducted at location B of the same mine. The hand pump was replaced by an air-hydraulic pump so as to obtain better control of the rate of loading. Also, larger circular plates of 8-, 10-, and 12-in. diameters were used in the tests. Table 2 gives the results obtained from these tests.

TABLE 2. - Bearing capacity tests on floor at location B, mine 1--circular plates

Bearing plate diameter, in.	Penetration rate, in./5 min.	Bearing pressure, psi	
		At initial failure	Maximum achieved
8.....	0.1	833	833
8.....	.1	598	598
8.....	.2	920	920
10.....	.2	1,122	1,122
10.....	.1	1,100	1,100
10.....	.1	661	661
12.....	.1	673	673
12.....	.1	414	414
12.....	.1	403	403

A total of 25 floor tests were thus made with circular bearing plates at location B in mine 1. The results of these tests are summarized in table 3. The relationship (fig. 1) between the maximum bearing pressure imposed during 4 in. of penetration and the area of the circular penetration plate was computed to be

$$P_{max} = 3{,}311 \text{ (bearing area)}^{-0.38} \quad (2)$$

or

$$\text{Failure load} = 2.837D^{1.28}, \quad (3)$$

where D is the diameter of the plate, and P_{max} = maximum bearing pressure = failure load/bearing area.

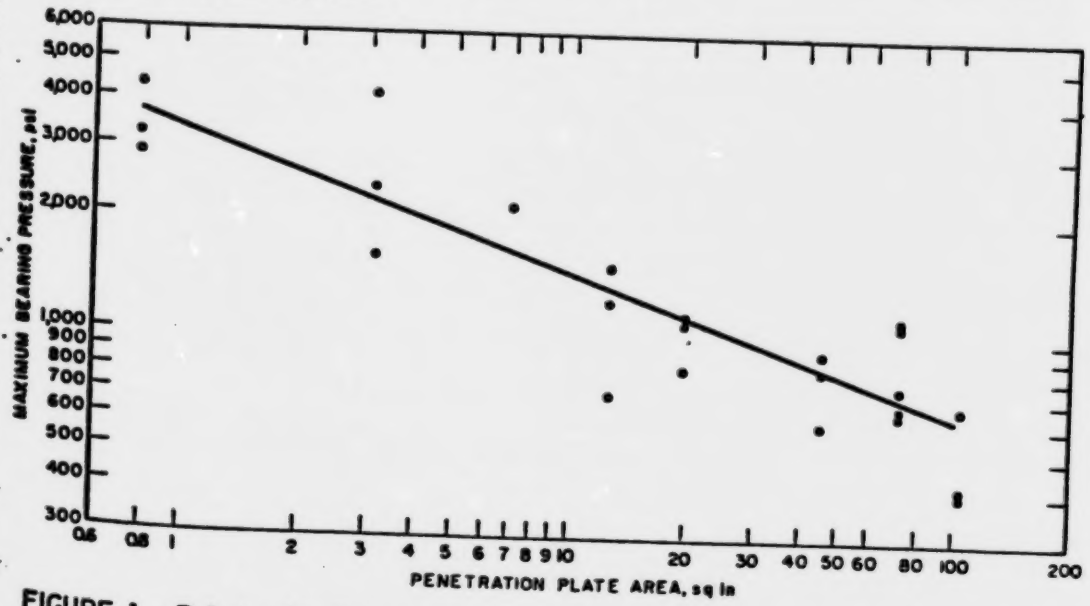


FIGURE 1. - Relationship Between Maximum Bearing Pressure During Total Penetration and Areas of Circular Penetration Plates, Mine I, Colorado.

TABLE 3. - Summary of bearing capacity tests on floor at location B, mine 1, Colorado--circular plates

Test No.	Plate diameter, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate (during test), tons	Bearing pressure at initial failure, psi	Maximum bearing pressure achieved, psi
CONSTANT RATE OF LOADING; SYSTEM CAPACITY = 30 TONS						
1	1	0.75	4.70	1.1	1,280	3,200
2	1	.5	3.66	1.25	1,920	2,880
3	1	1.5	3.78	1.7	4,050	4,320
4	2	2.5	2.20	2.5	1,600	1,600
5	2	3.8	4.72	3.8	2,400	2,400
6	2	6.6	4.34	6.6	4,200	4,200
7	3	7.5	4.38	7.5	2,130	2,130
8	4	7.8	4.08	7.8	1,240	1,240
9	4	5.0	3.94	9.4	800	1,500
10	4	4.4	3.60	4.4	700	700
11	5	8.2	3.98	8.2	830	830
12	5	6.3	4.08	10.6	640	1,090
13	5	11.1	4.05	11.1	1,130	1,130
14	10	No failure	.95	31.4	No failure	800
15	10	25.75	4.06	25.75	640	640
16	10	28.5	3.46	28.50	750	750
CONSTANT RATE OF PENETRATION; SYSTEM CAPACITY = 100 TONS						
17	8	18	4.08	18	833	833
18	8	13.4	4.00	13.4	598	598
19	8	20.6	4.01	20.6	920	920
20	10	39.7	4.40	39.7	1,100	1,100
21	10	23.9	4.00	23.9	661	661
22	10	40.5	4.04	40.5	1,122	1,122
23	12	34.9	3.93	34.9	673	673
24	12	21.5	4.00	21.5	414	414
25	12	20.9	4.03	20.9	403	403

Equations 2 and 3 can be written in the general form

$$P = K'A^{-m} \quad (4)$$

and

$$R = KD^{n} \quad (5)$$

where

P = bearing pressure at failure,

R = rupture or failure load,

A = area of bearing plate, and

m, n, K, K' = constants.

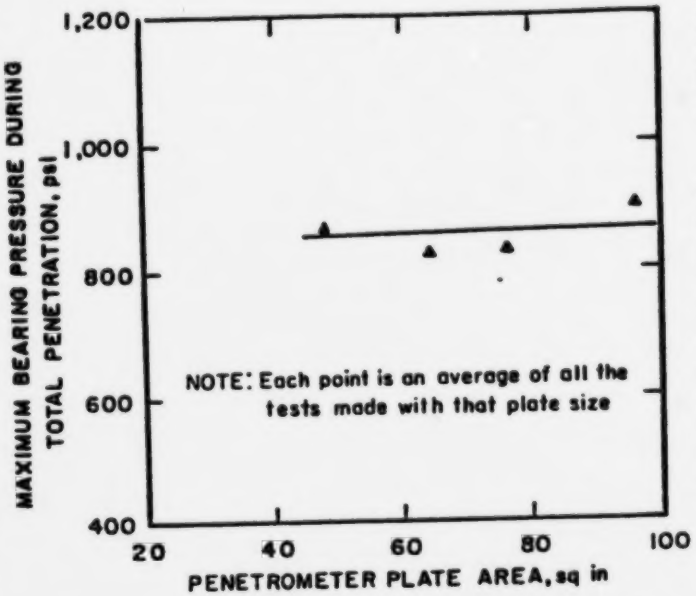


FIGURE 2. - Results of Bearing Capacity Tests at Colorado Mine With Rectangular Plates.

Equations 4 and 5 have been shown to hold true for different types of roof and floor strata in Europe by Jenkins (7-8) and Jahns (6).

There are extreme differences between the shapes and sizes of the circular bearing plates used in bearing capacity tests and the rectangular footplates and canopies of self-advancing longwall supports. Precautions must be used in extrapolating relationships similar to equations 2 and 3 to obtain the maximum allowable bearing pressure to be applied by the footplates and canopies.

It was therefore decided to reduce the differences between the size and shape of the bearing plate used in the tests and the prototype, by using square and rectangular bearing plates of sizes ranging from 6 by 6 in. to 12 by 12 in. and 6 by 18 in.

Results of tests made on the floor at location B in mine 1 with the rectangular bearing plates are given in table 4. Figure 2 is a plot of the average bearing capacity of the floor for a given plate versus the plate area.

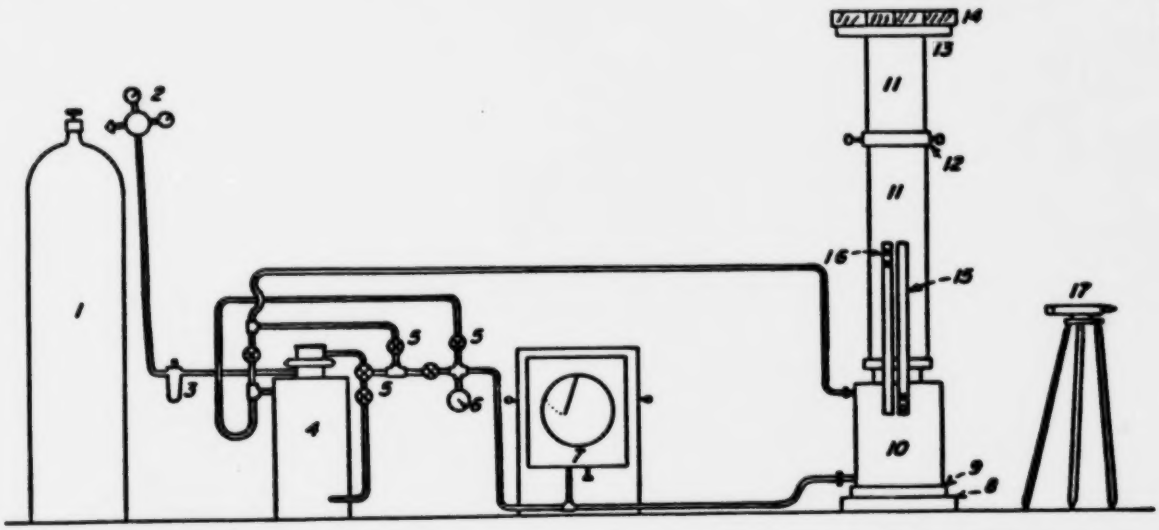
TABLE 4. - Bearing capacity tests on floor at mine 1, Colorado--rectangular plates

Test No.	Plate size, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate, tons	Bearing capacity at initial failure, psi	Maximum bearing pressure reached, psi	
						Individual test	Plate size average
1	6 x 12	27.7	3.97	27.7	1,157	1,157	-
2	6 x 12	18.5	3.92	18.5	775	775	864
3	6 x 12	14.8	4.00	14.8	619	661	-
4	6 x 18	24.5	4.00	24.5	638	638	-
5	6 x 18	39.5	4.00	39.5	1,031	1,031	835
6	6 x 18	32.0	4.00	32.0	835	835	-
7	8 x 12	38.0	3.98	38.0	1,180	1,180	-
8	8 x 12	19.3	4.00	19.3	600	600	824
9	8 x 12	22.3	4.02	22.3	692	692	-
10	8 x 16	54.0	4.00	54.0	1,113	1,113	-
11	8 x 16	53.5	4.00	53.5	1,113	1,113	908
12	8 x 16	23.6	4.00	23.6	452	497	-

In spite of the considerable scatter of individual tests, the plot indicates that the average bearing capacity per plate does not vary significantly with the size of the bearing plate. This differs from the relationships (equations 4 and 5)determined from tests with circular plates. It appears that an estimate of the average bearing capacity that is independent of the size of the bearing plate may therefore be obtained by averaging the bearing capacity values from tests made with as many of the rectangular plates as desired. The test was therefore modified to utilize only square and rectangular plates. The number of tests made to obtain reliable estimates of the lower limit of the bearing capacity was determined from the first three tests. If the average bearing capacity as determined by the first three tests was greater than 6,000 psi (capacity of the system using a 6- by 6-in. plate), no further tests were made. With lower average bearing capacity, correspondingly larger number of tests were made. Since the main purpose was to obtain a simple and efficient method of testing the bearing capacity of mine strata, the maximum number of tests made on any particular strata in a particular location was limited to a total of nine, made up of three tests each with three different plate sizes.

DESCRIPTION OF APPARATUS

The apparatus in its final form is shown schematically in figure 3 and includes:



1 Nitrogen cylinder
2 Air pressure regulator connected to compressed air or N_2 cylinder
3 Air-line lubricator
4 1:150 air-hydraulic pump
5 High-pressure valves
6 20,000-psi pressure gage
7 10,000-psi pressure recorder
8 Penetration plate
9 Adapter
10 100-ton ram
11 Tubular steel extension
12 Bolted sleeve
13 Bearing plate 14in x 14in
14 Hardwood blocking
15 Floor penetration scale
16 Roof penetration scale
17 Sighting telescope

FIGURE 3. - Schematic of Bearing Capacity Test Apparatus.

a. A 100-ton-capacity hydraulic ram with a stroke of about 10 in.

b. An air-hydraulic pump with an air to fluid pressure ratio of 1 to 150, capable of delivering fluid pressures beyond the ram capacity and capable of maintaining constant fluid pressure for selected periods of time.

c. A 0- to 10,000-psi clock-wound continuous pressure recorder with a chart rotation time of 4 hours.

d. A level telescope or equivalent sighting device to measure strata penetration and ram extension remotely by observing movement of separately attached scales having 0.02-in. divisions.

e. A nitrogen cylinder (2,000 psi) or compressed-air supply.

f. An air regulator capable of controlling air pressure from 0 to 100 psi.

g. A set of rectangular penetrometer plates of dimensions as shown in figure 4 and table 5. The plates are of 2-in. thickness to minimize distortion under load. Edges of the plates are rounded to minimize stress concentration.

h. A 14- by 14-in. steel bearing plate with accessory hardwood blocking is used such that a bearing area greater than the penetrometer plate being used is provided at the end opposite to the penetrometer plate.

i. Miscellaneous parts, including high-strength steel tubular sectionalized extensions, high-pressure hydraulic lines, high-pressure valves and fittings, and construction tools required for preparation of test sites.

Figure 5 is a view of the apparatus prior to the bearing capacity test in a mine.

TEST PROCEDURE

The following is a description of the final procedure adopted to assure uniformity of all bearing capacity tests.

Selection of Test Sites

Each test site was in a development entry in that portion of the mine where a longwall was operating or where a longwalling operation was being considered. The test sites appeared to be representative of the conditions normally present in the mine. This precluded tests under the following conditions:

a. Abnormal quantities of water.

b. Fractured bottom or top.

c. Local anomalies such as faulted areas and pinch-outs.

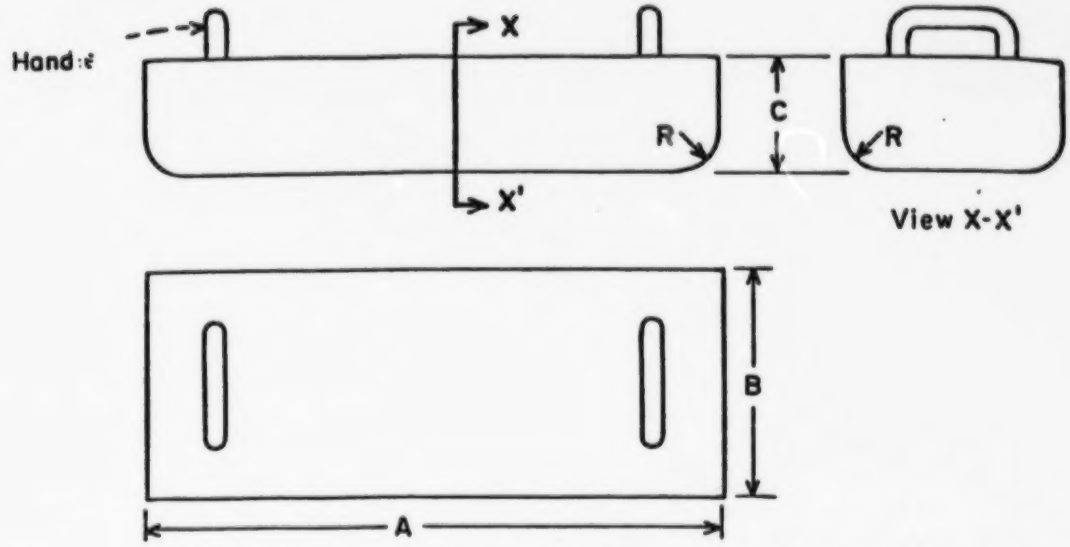


FIGURE 4. - Penetrometer Bearing Plate Dimensions. (See table 5.)

TABLE 5. - Penetrometer bearing plate dimensions

Plate No.	Dimension, in.				Area, in^2
	A	B	C	R	
1.....	6	6	2	3/8	31.64
2.....	9	6	2	3/4	34.73
3.....	12	6	2	3/4	47.81
4.....	18	6	2	3/4	76.60
5.....	8	8	2	3/8	56.72
6.....	12	8	2	3/4	64.44
7.....	16	8	2	3/4	96.06
8.....	10	10	2	3/8	70.28
9.....	12	12	2	3/8	132.25

Information regarding the character and thicknesses of the immediate roof and floor strata was obtained. Occasionally, it was necessary to hand drill the roof and floor to obtain this information.

Preparations for Test

The floor and roof were cleaned of loose material, and the test surfaces were made flat to achieve reasonably uniform loading. The bearing area of the assembly at the end opposite the penetrometer plate was made much larger than the penetrometer plate through the use of the large steel bearing plate and hardwood blocks. Wooden wedges were used to provide full contact of the blocking with the roof.

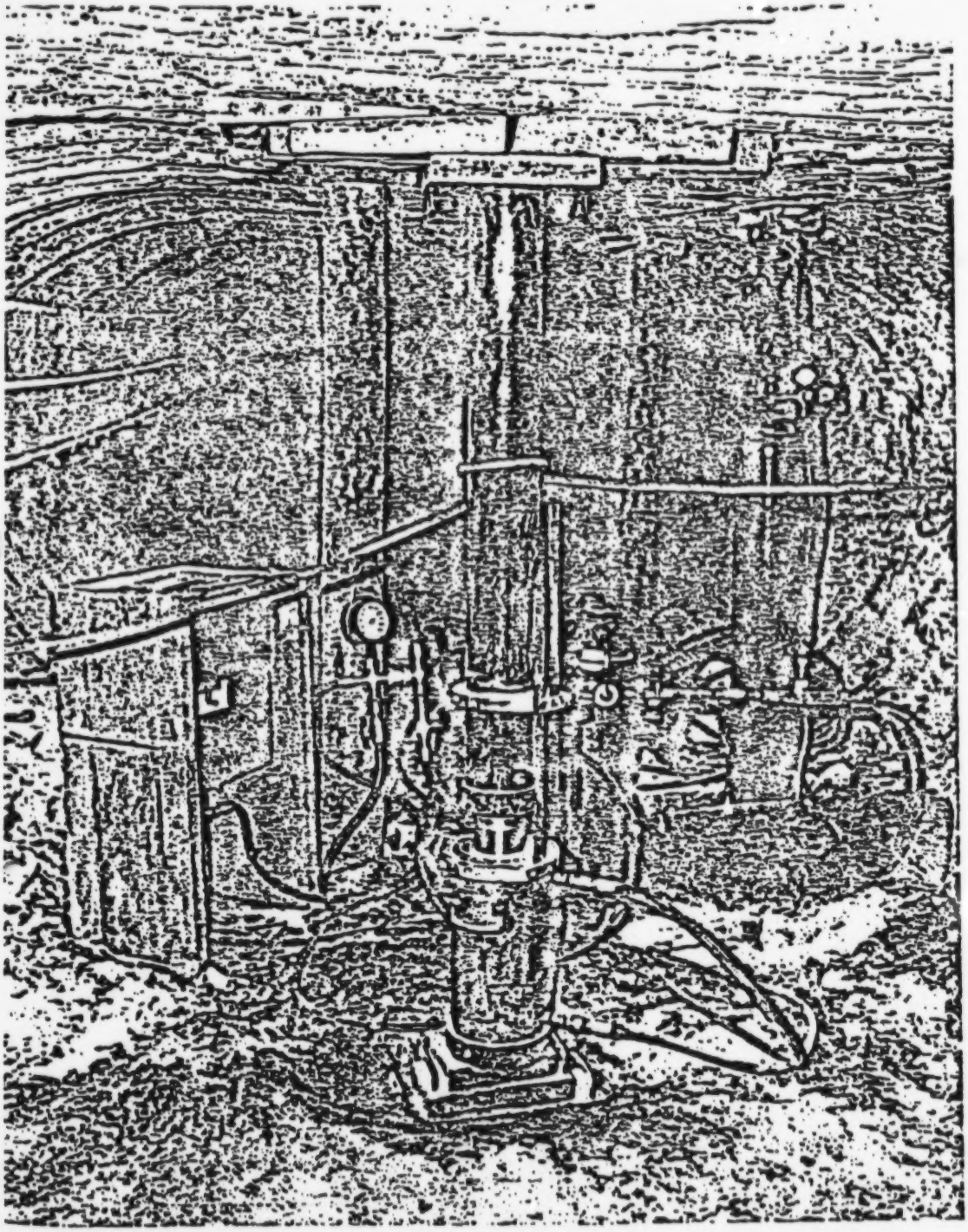


FIGURE 5. - Apparatus Used To Test the Bearing Capacity of Mine Roof and Floor.

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The graduated scales were attached separately to the ram and the casing so as to permit measurement of penetration of the bearing plate into the strata and deformation of the hardwood blocking. The level telescope was set up about 10 ft away from the assembly so that it was outside the influence of possible strata movements caused by the test. It was sighted on the two graduated scales. The line of sight through the telescope served as an independent datum against which the movements of the scales were noted.

Procedure

The system was assembled as shown in figure 3. The load on the selected penetrometer plate was increased by operation of the air-hydraulic pump. The rate of loading was controlled by manipulation of the air-line regulator such that a constant rate of penetration of 0.2 in. per 5-minute interval was maintained. The tests were terminated when either a 4-in. penetration of the plate into the strata or the load capacity of the ram (100 tons) was reached. If the strata failed, the load at initial failure was noted. Failure of the strata was determined by that bearing pressure between the plate and the rock at which the penetration rate increased noticeably while being accompanied by a drop in the recorded fluid pressure. It was indirectly determined by the maximum pressure in the recorded plot of fluid pressure versus time. A

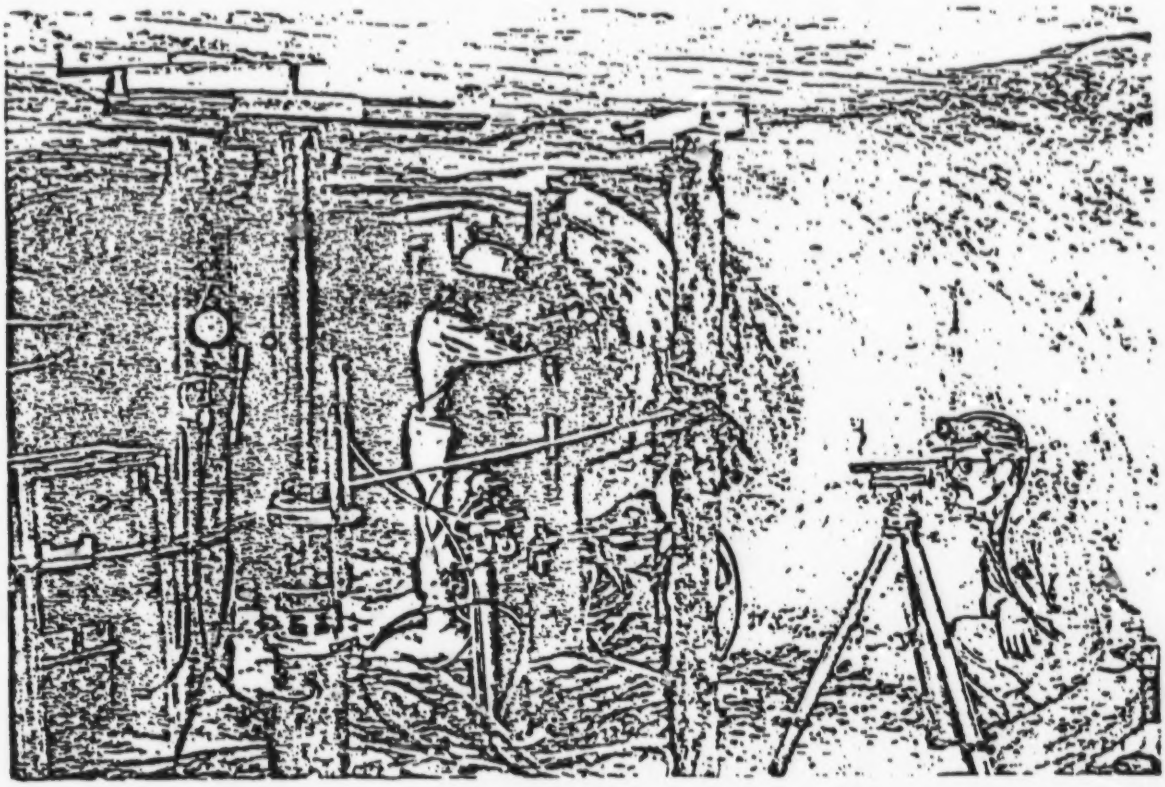

FIGURE 6. - Bearing Capacity Test in Progress at Mine I.

constant rate of penetration was maintained even after failure. In cases where layers of the strata failed independently, bearing pressures at successive failures of the strata were also noted.

The penetrometer plate selected for the first test was usually of the largest size that could reasonably be expected to produce failure at loads of less than 100 tons. The sequence of plate sizes used in successive tests were randomized, and the spacing between test points was always more than 5 ft.

Figure 6 shows the bearing capacity test apparatus being used to test the floor in a coal mine.

DESCRIPTION OF FIELD TESTS AND ANALYSIS OF RESULTS

Tests of the roof and floor were made at eight other mines, located in seven States, selected as being representative of coal measures and of operating longwalls in the United States. The results are given in table 6 and are summarized in table 7. Figure 7 gives the stratigraphy of the immediate roof and floor in the test areas of each of the mines where bearing capacity tests were made.

In mines where the roof and/or floor strata were strong, the strata did not fail at the capacity of the system (100 tons) even when the smallest bearing plate (6 by 6 in.) was used, hence the average bearing capacity exceeds 6,000 psi, the maximum bearing pressure applied by our system. The roofs of mines 2, 3, 4, 7, and 9 and the floors of mines 2 and 4 fall in this category.

In other mines, as in the floors of mine 3, mine 8, and mine 9 and the roof of mine 5, the results indicate that the bearing capacity depends upon the strength of the immediate layer tested and varies, for example, with the thickness of the coal in the floor or the thickness of draw slate in the immediate roof. In mines with similar strata, it is suggested that sufficient tests be made to obtain reliable estimates of bearing capacity for the different layers of the roof and/or floor strata. It may then be possible to determine, for example, the optimum thickness of floor coal to be left such that the floor remains competent under operating conditions.

Tests made at mine 8 indicate that the bearing capacity of the fireclay layer, beneath the 4- to 8-in. thickness of shale that comprises the floor of the longwall panel, is much greater than that of the shale layer. If the shale proved to be incompetent and allowed penetration of the longwall supports into the floor under operating conditions, it might prove to be advantageous from a ground control standpoint to include the shale layer in the cut made by the mining machine and to use the fireclay as a competent floor. Results of tests made at mine 9 show a similar situation, with indications being that the shale layer below the fireclay and coal in the floor is the strongest of the three layers. However, in this mine, the longwall floor has a 4-in. layer of hard pan over the 4-in. layer of coal and apparently this hard pan layer contributes to a competent floor.

TABLE 6. - Bearing capacity tests, mines 2-9

Test	Plate size, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate, tons	Bearing capacity at initial failure, psi	Maximum bearing pressure, psi	Remarks
MINE 2, UTAH							
Floor:							
1	6 x 9	No failure	0.07	107.5	>6,205	6,205	Shale floor.
2	6 x 6	do.	.10	105.5	>6,675	6,675	Do.
3	6 x 6	do.	.20	106.5	>6,743	6,743	Do.
4	6 x 6	do.	.10	104.5	>6,607	6,607	Do.
Roof:							
1	6 x 6	do.	.42	107.5	>6,810	6,810	Shale roof.
2	6 x 6	do.	.36	107.5	>6,810	6,810	Do.
3	6 x 6	do.	.36	107.5	>6,810	6,810	Do.
MINE 3, ILLINOIS							
Floor:							
1	6 x 6	89.5	2.51	89.5	5,660	5,660	Coal floor; 20 in. of coal.
2	8 x 8	84	4.10	84	2,970	2,970	14 in. of coal.
3	8 x 8	No failure	.50	103	>3,630	3,630	Over 24 in. of coal.
4	6 x 9	do.	.58	101.5	>5,850	5,850	Do.
5	10 x 10	do.	.63	104	>2,870	2,870	Do.
6	6 x 6	do.	.60	105	>6,640	6,640	Do.
7	6 x 6	78	2.48	82.5	4,930	5,280	Do.
8	6 x 9	No failure	.29	104.5	>6,020	6,020	Do.
Roof:							
1	6 x 6	84	3.54	84	5,320	5,320	Shale roof.
2	6 x 9	No failure	.41	101.5	>5,860	5,860	Do.
3	6 x 6	do.	.47	103	>6,500	6,500	Do.
4	6 x 9	do.	.33	104	>6,000	6,000	Do.
MINE 4, VIRGINIA							
Floor:							
1	6 x 6	81	3.95	81	5,145	5,145	Hard fireclay floor.
2	8 x 8	No failure	.20	108	>3,800	3,800	Do.
3	6 x 9	do.	.19	110	>6,340	6,340	Do.
4	6 x 6	do.	.10	111	>7,000	7,000	Do.
5	6 x 6	do.	.10	109	>6,950	6,950	Do.
Roof:							
6	6 x 6	do.	.34	108	>6,800	6,800	Shale roof.
7	6 x 6	do.	1.03	110	>6,975	6,975	Do.
8	6 x 6	do.	1.09	109	>6,900	6,900	Do.
MINE 5, WEST VIRGINIA							
Floor:							
9	6 x 6	38	4.00	55	2,390	3,480	Shale/fireclay floor.
10	8 x 8	No failure	.65	109	>3,855	3,855	Do.
11	6 x 6	60	3.59	60	3,786	3,786	Do.
12	6 x 9	62	3.94	62	3,580	3,580	Do.
13	6 x 9	59	3.99	59	3,430	3,430	Do.
14	6 x 6	26	3.87	26	1,612	1,612	Do.
15	8 x 8	104	1.86	104	3,666	3,666	Do.
16	6 x 9	82	2.51	82	4,720	4,720	Do.
17	8 x 8	79	3.36	79	2,780	2,780	Do.
Roof:							
1	6 x 6	45	4.00	45	2,835	2,835	20 in. of draw slate.
2	10 x 10	No failure	.52	109	>3,070	3,070	30 in. of draw slate.
3	6 x 18	do.	.37	109	>2,840	2,840	Do.
4	6 x 12	do.	.30	107	>4,500	4,500	28 in. of draw slate.

TABLE 6. - Bearing capacity tests, mines 2-9--Continued

Test	Plate size, in.	Load on plate at initial failure, tons	Total penetration, in.	Maximum load on plate, tons	Bearing capacity at initial failure, psi	Maximum bearing pressure, psi	Remarks
MINE 5, WEST VIRGINIA--Continued							
Roof: (Con.)							
5	6 x 6	No failure	0.46	104	>6,600	6,600	30 in. of draw slate.
6	6 x 6	do.	.45	104	>6,600	6,600	Do.
7	6 x 6	60	4.00	70	3,810	4,398	Do.
8	6 x 6	No failure	.37	107	>6,776	6,776	Do.
MINE 6, WEST VIRGINIA							
Floor:							
4	6 x 6	88	3.48	88	5,552	5,552	Shale floor.
5	8 x 8	79	3.50	79	2,794	2,794	Do.
6	8 x 8	88	3.94	88	3,115	3,115	Do.
7	8 x 8	77	4.00	77	2,733	2,733	Do.
8	6 x 6	42	4.00	42	2,632	2,632	Do.
9	6 x 6	63	4.02	63	3,990	3,990	Do.
Roof:							
1	6 x 6	67	3.94	72	4,260	4,570	Shale roof.
2	6 x 6	No failure	.32	103	>6,530	6,530	Do.
3	6 x 6	do.	.19	105	>6,640	6,640	Do.
MINE 7, WEST VIRGINIA							
Floor:							
4	6 x 6	54	3.74	54	3,447	3,447	On shale.
5	8 x 8	No failure	2.16	104	>3,680	>3,680	Verge of failure.
6	8 x 8	102	3.34	102	3,590	3,590	On shale.
7	6 x 6	45	4.12	45	2,835	2,835	Do.
8	8 x 8	99	4.12	99	3,490	3,490	Do.
9	6 x 6	57	4.10	57	3,610	3,610	Do.
Roof:							
1	6 x 6	No failure	.84	104	>6,572	6,572	On coal top.
2	6 x 6	do.	1.34	105	>6,670	6,670	Do.
3	6 x 6	do.	.51	104	>6,600	6,600	Do.
MINE 8, PENNSYLVANIA							
Floor:							
1	6 x 6	62	4.00	62	3,922	3,922	Shale floor.
2	8 x 8	49	3.15	49	3,107	3,107	2 in. of floor coal.
3	6 x 6	No failure	.36	97	>6,164	6,164	Hard fireclay floor.
4	6 x 6	do.	.54	102	>6,436	6,436	Do.
8	6 x 6	do.	.64	102	>6,436	6,436	Do.
Roof:							
5	6 x 6	do.	.28	95	>6,028	6,028	4 in. of top coal.
6	6 x 6	79	4.00	79	5,009	5,009	Do.
7	6 x 6	No failure	.20	102	>6,436	6,436	Sandy shale roof.
MINE 9, PENNSYLVANIA							
Floor:							
1	6 x 6	40	3.19	40	2,563	2,563	Coal floor.
2	8 x 8	77	4.00	77	2,730	2,730	Do.
3	8 x 8	44	3.94	44	2,800	2,800	Do.
4	6 x 6	32	3.90	37	2,020	2,360	Do.
5	8 x 8	60	3.16	107	2,112	3,780	Fireclay floor.
6	6 x 6	81	4.10	81	5,145	5,145	Shale floor.
Roof:							
7	6 x 6	No failure	.23	98	>6,232	6,232	Shale roof.
8	6 x 6	do.	.17	103	>6,504	6,504	Do.
9	6 x 6	do.	.80	103	>6,530	6,530	Do.

TABLE 7. - Summary of bearing capacity test results

Mine No.	1	2	3	4	5[1]
Area	Colorado	Utah	Illinois	Virginia	West Virginia
Cover, ft	350	1,000-2,000	660	1,350-2,000	500-800
Seam thickness, in.	108-144	65-80	108	52	50
Height of extraction, in.	84	82	84	52	50
Test location	4 S off main W	16 left	3 N	#4 development entry-#2 plow	1st right
Roof geology	Coal	Sandy shale	Shale	Sandstone over 18 to 72 in. shale	18 to 60 in. draw slate up to 4 in. rider seam
Roof, average bearing capacity, psi	>3,000	>6,000	>5,000	>6,000	>5,000
Floor geology	Shale	Shale	2 ft coal over fireclay	Hard fireclay	6 in. shale over fireclay
Floor, average bearing capacity, psi	800	>6,000	>5,000	>6,000	3,400

Mine No.	6	7[2]	8	9[3]
Area	West Virginia	West Virginia	Pennsylvania	Pennsylvania
Cover, ft	400-800	400-600	440-600	500-800
Seam thickness, in.	60	80	58	54
Height of extraction, in.	60	72	54	50
Test location	11 butt left	6 right off 7 N mains	Face area	D west
Roof geology	Shale	Draw slate over 8 in. top coal	4 in. top coal, then sandy shales	Shale
Roof, average bearing capacity, psi	>4,500	>6,500	>5,000	>6,000
Floor geology	Shale	Shale	6 in. shale over 24 in. fireclay over sandstone	6 in. coal over 4 in. fireclay over shale
Floor, average bearing capacity, psi	3,400	3,400	Shale floor = 3,500; fireclay floor >6,000	Coal floor = 2,700; shale floor >3,800

[1]Strength of roof varies directly as thickness of draw slate.
[2]6 in. bottom coal left on longwall floor.
[3]Longwall floor has 6 in. layer of hard pan over coal.

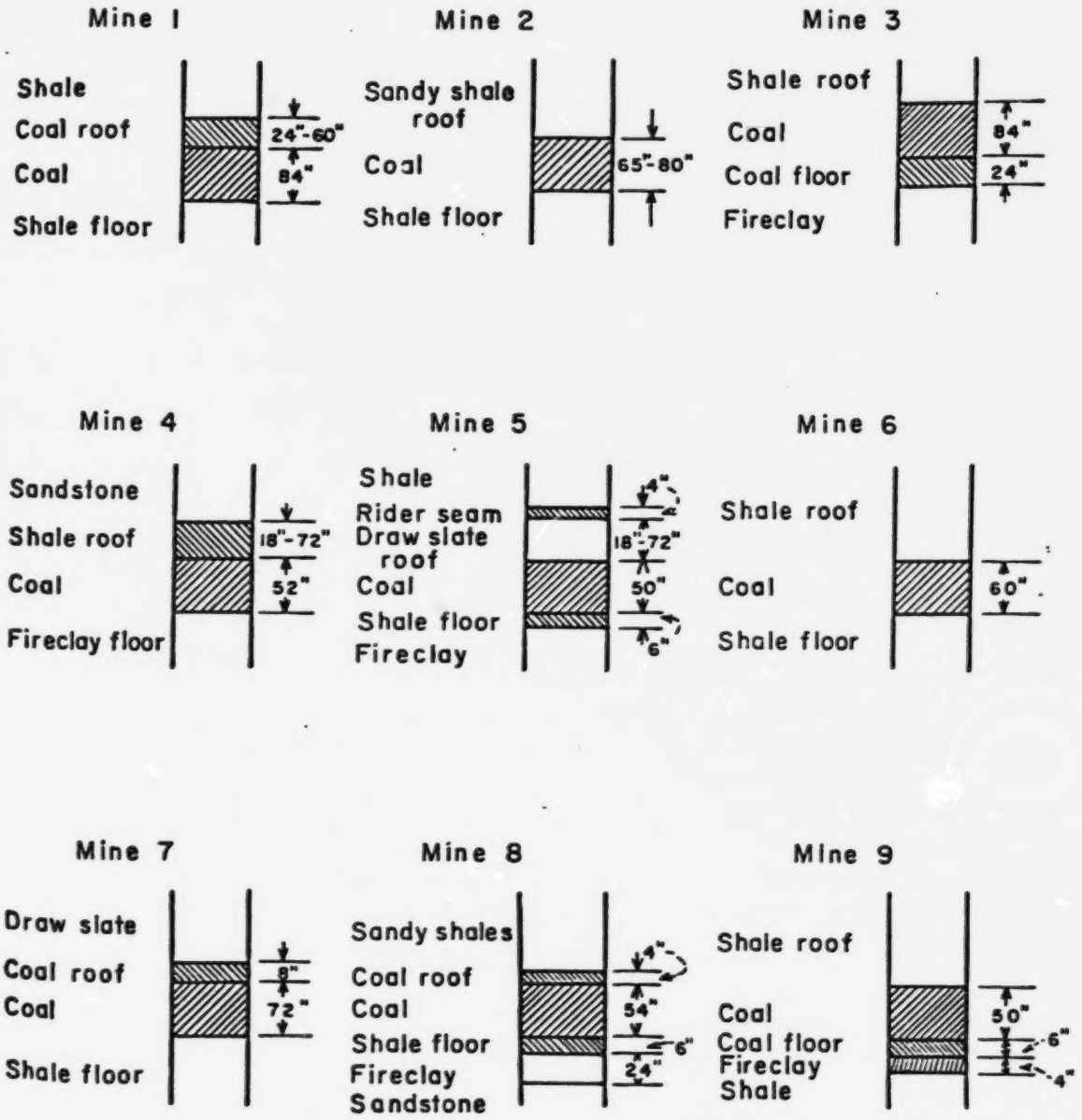

FIGURE 7. - Stratigraphy of Immediate Roof and Floor in Test Areas.

Mines 2, 4, 5, 7, 8, and 9 had operating longwalls at the time the tests were conducted. In each of these mines the estimates of the lower limit of the bearing capacity of the roof and the floor as determined by our tests were compared with the maximum bearing pressure applied by the hydraulic support on the longwall as obtained from the manufacturer's specifications. In each case the estimate of the lower limit of the bearing capacity was found to be much greater than the maximum bearing pressure of the support. None of the

longwalls had any unusual difficulty with supports punching into the floor, which indicates that the lower limit of bearing capacity of mine strata as estimated by the bearing tests may be used to determine the maximum allowable bearing pressure for a support. Additional tests conducted at mines that have poor floor/roof will provide complete proof of the reliability of this bearing capacity test. Future tests are being considered in mines having roof and floor of questionable bearing capacity with reference to the bearing pressures applied by longwall supports.

In analyzing the data obtained from testing in the nine mines, the lowest bearing capacity value as obtained from the test in each mine has been taken as an estimate of the lower limit of bearing capacity of the floor/roof in that mine. These values are tabulated in table 8; for comparison we have also included the design value of bearing pressure at yield load, for the support used in each mine.

TABLE 8. - Bearing capacity: Average and lower limits

Mine	Floor			Roof		
	Average bearing capacity, psi	Estimate of lower limit, psi	Design bearing pressure of support in use at yield, psi	Average bearing capacity, psi	Estimate of lower limit, psi	Design bearing pressure of support in use at yield, psi
1	858	490	No longwall	-	-	-
2	>6,000	>5,000	400	>6,000	>5,000	200
3	>4,820	2,970	No longwall	>5,000	>5,000	No longwall
4	>5,000	>4,000	780	>6,000	>5,000	100
5	3,420	1,600	No longwall	>4,600	>2,800	No longwall
6	3,470	2,600	780	>5,000	>4,000	100
7	3,440	2,800	800	>6,000	>5000	500
8	5,000	3,100	510	>5,000	>4,000	360
9	2,890	2,000	200	>6,000	>5,000	100

DISCUSSION AND CONCLUSIONS

Results obtained from bearing capacity tests with circular plates agree with prior work done by Jenkins and Jahns. The accuracy of estimating allowable bearing pressures for rectangular longwall base plates and canopies is improved by carrying out tests with square and rectangular bearing plates of large areas, in order to minimize extrapolation. The average and the lower limit of bearing capacity of the strata as obtained from the tests may then be used to determine the maximum allowable bearing pressure for a chock of a specified yield load.

The tests in mine 1 show that the bearing capacity of rock varies according to the size and shape of the bearing plates. The tests also indicate that the rate of change of bearing capacity decreases as the bearing plate diameter is increased. The average bearing capacity of a rock layer tends to become a constant when tests are made with plates of large areas.

Owing to the natural inhomogeneity of strata found in the mining environment, the scatter of the results obtained from the bearing capacity tests is rather pronounced. It is evident that the results obtained are a function of the strength of the strata and also of the test procedure. We have attempted to minimize the latter variable by following a standard test procedure. The average bearing capacity obtained from our tests is thus an estimate of the true average bearing strength of the rock.

The estimate of the lower limit of bearing capacity as determined by the lowest value of the bearing capacity tests made on the roof/floor of a mine may be used to determine the maximum allowable bearing pressure that may be applied by a longwall support on the roof/floor of that mine. Before making a final decision about introducing a longwall support system, bearing tests should be made to determine the capacity of the strata to support load over a period of several days. Such tests would simulate conditions such as the standing of a wall over a weekend.

The bearing strength of the strata may be considerably weakened by water being made in the longwall area and it may be considerably strengthened through selective mining, such as leaving a 12-in.-thick layer of top or floor coal. For proper interpretation of the tests, the tests should be made under essentially similar conditions to those under which the longwall will be operated.

Most of the operators of longwalls in the United States appear to have a good idea of the strength of the roof and floor strata with respect to the bearing pressures applied by their support equipment. This knowledge has come through experience and trial and error. The merits of doing bearing capacity tests lie in that knowledge of the strength of the rock is obtained before selecting and installing longwall equipment, which will help to avoid costly mistakes and delays for both the operator and the equipment manufacturer.

The test procedure and the test results presented herein show that a convenient, effective, and reliable test to determine the bearing capacity of mine roof and/or floor has been developed. The apparatus can be made quite easily by any mining company; the tests can be made by untrained personnel working under some engineering supervision; the results of the tests may be used without any questionable extrapolation, and the expense involved is minimal.

Determination of the bearing capacity of the mine strata in the vicinity of a proposed longwall prior to selection of support equipment may be well worth the effort when the dangers, the difficulties, and the expenses that occur due to the selection of inappropriate support systems are considered.

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INT.-BU.OF MINES,PGH.,PA. 14984

TABLE PLATE-LOAD TESTS RESULTS

	Test No	Dia 10^{-3} m (in)	yield q_p MN/m² (lb/in²)	ult. q MN/m² (lb/in²)	q_p MN/m² (lb/in²)	$w/2a$ %	E_d MN/m² (lb/in²)	d/a	r/a	$L/2a$	$l/2a$	L/l
Wilson Seam	1	101·6 (4)	3·03 (440)	4·14 (600)	4·14 (600)	22·6	17·73 (2570)					
	2	152·4 (6)	1·34 (195)	1·93 (280)	1·93 (280)	15·0	7·93 (1150)					
	3	76·2 (3)	4·48 (650)	5·38 (780)	5·38 (780)	26·6	19·38 (2810)					
	4	101·6 (4)	3·66 (530)	5·17 (750)		15·0	33·52 (4860)					
	5	127·0 (5)	5·04 (730)	6·21 (900)	6·21 (900)	16·4	41·94 (6080)					
4ft seam	6	76·2 (3)	15·86 (2300)	16·17 (2344)	16·17 (2344)	8·3	106·22 (15400)	0·92	1·50	2·34	1·50	1·55
	7	88·9 (3½)	15·86 (2300)	19·80 (2870)	15·04 (2180)	16·3	102·70 (14890)	0·79	1·35	3·38	1·80	1·87
	8	101·6 (4)	8·97 (1300)	9·09 (1318)	6·97 (1010)	8·0	68·08 (9870)	1·00	1·25			
	9	114·3 (4½)	10·35 (1500)	12·52 (1815)	11·04 (1600)	16·7	79·25 (11490)	0·61	1·40	2·88	2·45	1·18
	10	76·2 (3)	19·17 (2780)	19·38 (2810)	17·24 (2500)	10·0	148·98 (21600)	0·92	1·57	3·16	2·23	1·41
	11	101·6 (4)	15·11 (2190)	18·18 (2636)	18·18 (2636)	14·2	123·88 (17960)	0·59	1·23	2·98	2·56	1·15
	12	76·2 (3)	6·55 (950)	7·04 (1020)	9·16 (1328)	18·0	17·31 (2510)	1·44	1·70	3·94	3·15	1·25
	13	88·9 (3½)	10·29 (1492)	11·88 (1722)	7·93 (1150)	11·7	78·70 (11410)	0·90	1·35	2·59	2·25	1·15
	14	101·6 (4)	9·10 (1320)	9·10 (1320)	12·12 (1757)	12·5	35·31 (5120)	1·00	1·28	1·97	1·97	1·00
	15	114·3 (4½)	9·93 (1440)	12·93 (1875)	11·49 (1666)	9·2	141·12 (20460)		1·14			

q_p = proportional limit
q = bearing capacity at failure
q_u = ultimate bearing capacity
w = deflection at failure
E_d = modulus of deformation
$2a$ = diameter of the plate
d = depth of the crushed bulb
r = radius of the crushed bulb
L = maximum length of the disturbed area
l = minimum length of the disturbed area

* In tests 12 and 14, q_p is $> q$. As in the other tests in this group a peak value, q, was reached and followed by a pressure drop. However in these tests, rather than stabilizing as illustrated in Fig. 3, section IV, the pressure again built up before stabilizing at a value $q_p > q$.

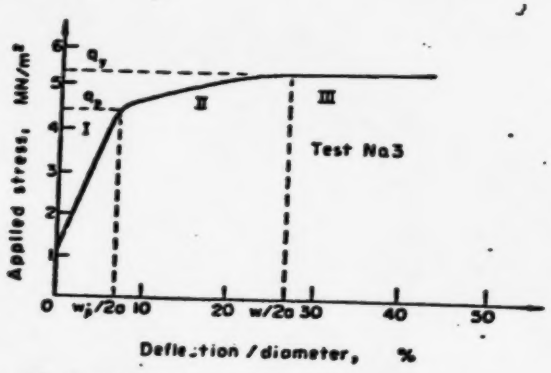


Fig. 2. Yielding failure of an "elastic-plastic" substance.

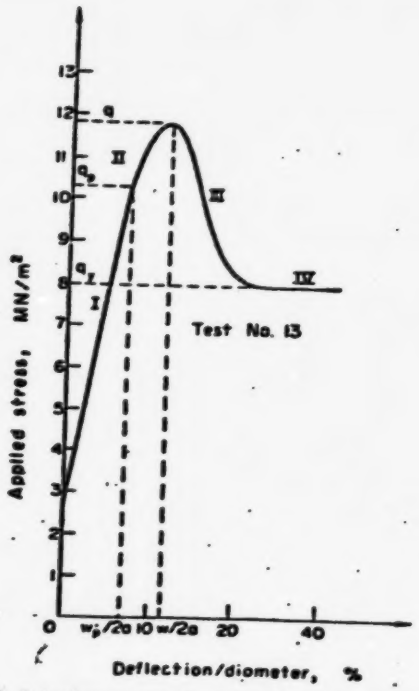


Fig. 3. Behaviour of an "elastic-brittle-plastic" substance.

APPENDIX 4-2

VOLTAGE REGULATION CALCULATIONS AND RECOMMENDATIONS
McINTYRE MINES SMOKY RIVER MINE
SERVICE MACHINE

VOLTAGE REGULATION CALCULATIONS AND RECOMMENDATIONS

McINTYRE MINES
SMOKY RIVER MINE

November 1982

Objective

To predict the voltage regulation at key points in the power distribution system under various operational conditions, to identify any critical locations and recommend modifications where necessary to ensure conformity with an acceptable performance standard.

Basic Assumptions

The principal data is contained in Mr. Howard Owen's letter of 20 October 1982 and the drawing which accompanied that letter. Mr. Klaxton subsequently confirmed that the surface transformer was 25/7.2 kV, 3.75 MVA with a 5.7% impedance value. There is no automatic voltage regulation and no power factor correction for this circuit.

The motor data used for the 250 HP. AFC motors was for a Reliance motor commonly used in the USA (performance curve 609347-22). This motor has a locked-rotor current of 7.19 x FLC and a pull out torque of 2.59 x FLT.

It was also assumed that the transformers supplying other sections of the mine were loaded to two thirds of their capacity at a power factor of 0.75.

Summary of Recommendations

The capacity of the transformer supplying the Shearer should be increased from 500 kVA to 1,000 kVA. This is necessary in order to reduce the excessive voltage drop at the motor controls on starting.

The 2/0 trailing cable size should be retained for the A.F.C. tail end motor, since there is very little to be gained by going to a larger size.

The trailing cable size for the Shearer could be reduced from 3/0 to 2/0 with negligible effect on the performance of the Shearer. This would make the AFC tail end motor cable identical to the Shearer cable which would allow economies in the stock holding of spare cables.

Longer term recommendations

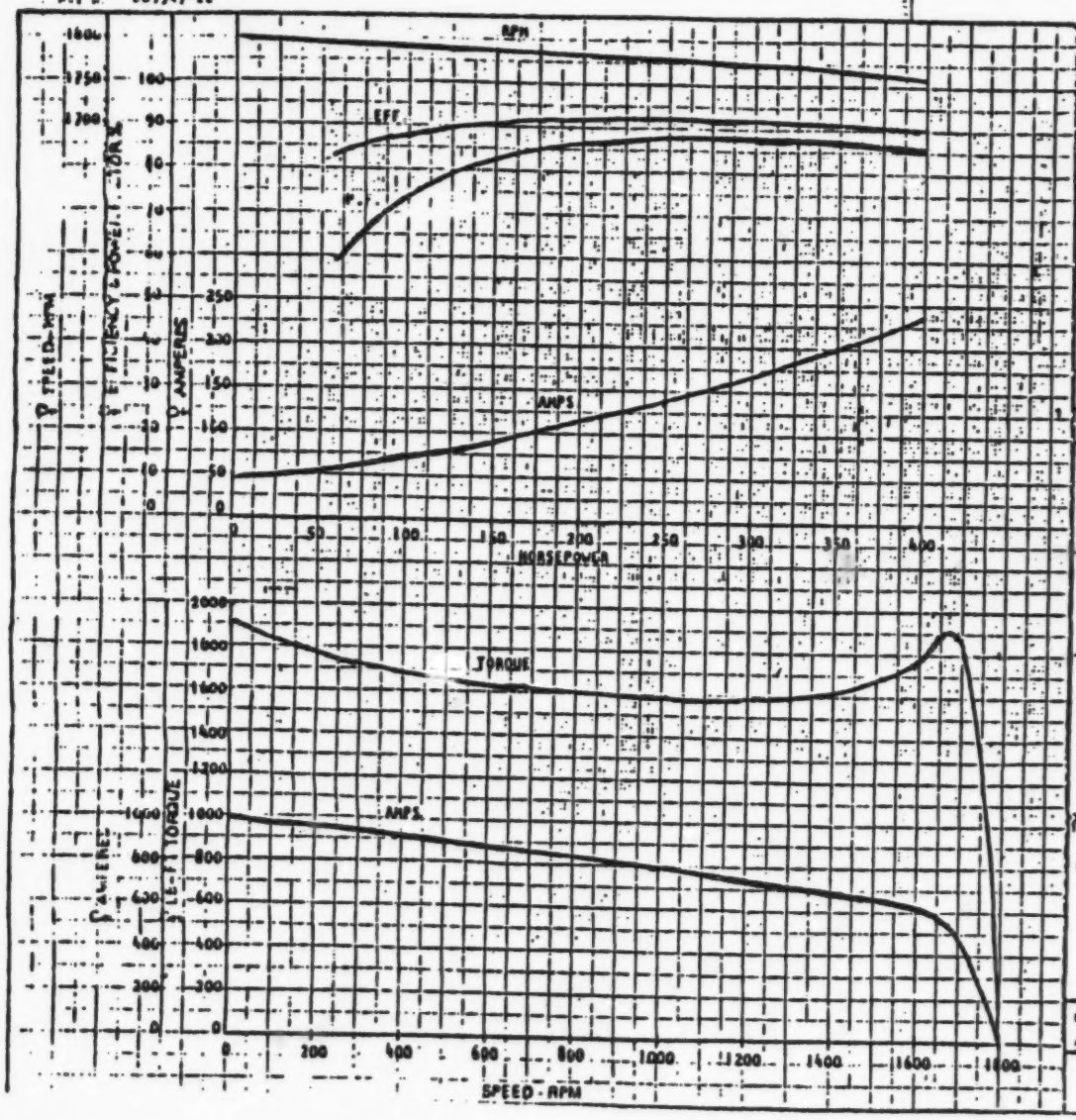
The use of fixed transformer taps does allow for some compensation for voltage drop but it does nothing to help voltage regulation. High voltages under light load conditions can be a serious problem.

Although conventional automatic tap changing transformers cannot compensate for the short term load variations caused by the starting of induction motors, they are able to give compensation for that portion of the voltage drop attributable to the base load of the system. In a suitably designed system an automatic tap changing transformer located on the surface can be adjusted to maintain the correct average voltage at an underground substation. An installation of this type can eliminate about one third of the total voltage regulation problem.

Power factor correction is usually economically viable simply from the savings in electricity charges. This can be achieved by capacitors or synchronous motors on the surface. Where it is practicable to locate capacitors closer to motor loads additional benefit can be gained. This benefit can be translated into reduced voltage regulation or a reduction in transformer or cable sizes.

An inbye transformer equipped with a high speed voltage booster with solid state controls could operate with sufficient speed to correct most of the short-term voltage regulation caused by motor starting currents.

A combination of this high speed booster underground with a conventional automatic tap changer at the surface would reduce voltage regulation to very small proportions.



NOMINAL RATING

FRAME	ENCLOSURE	H.P.	RPM
EBX447T	FCXP	250	1780
AMPERES 240/139	VOLTS 550/950	PHASE/HERTZ 3/60	
DUTY CONT.			

MOTOR NUMBER: P44G361A

MAXIMUM RATING

TIME	H.P.	°C RISE BY RESISTANCE	°C AMBIENT TEMP.

WINDING DATA

REF	S/S	VOLTS	RES. AT 20°C
	533161	550/950	.027/.081

ROTOR RELIANCE NO. 414957-36-EE
ROTOR WR² 60.5 LB-FT²
AIR GAP PER SIDE .055

NOTE 1: WINDINGS FOR VOLTAGES OTHER THAN NOMINAL WILL PRODUCE EQUIVALENT SPEED AND TORQUE.

NOTE 2: AMPERES AT VOLTAGES OTHER THAN LISTED ON CURVE ARE INVERSELY PROPORTIONAL TO THE VOLTAGE.

NOTE 3: INDICATED TORQUE AND AMPERES ARE NOMINAL VALUES AT RATED VOLTAGE. MAXIMUM VARIATIONS OF ±10% MAY BE EXPECTED AT NOMINAL LOADS AND ±10% AT LOCKED ROTOR.

NOTE 4: ALL PERFORMANCE CURVE DATA TAKEN WITH 550 VOLTS APPLIED AT MOTOR TERMINALS.

NOTE 5: ALL PERFORMANCE CURVE DATA TAKEN AT 105°C RISE BY RESISTANCE.

DRAWN BY G.R. KARLAK DATE 10-23-75
APPROVED DATE

A-C MOTOR PERFORMANCE CURVES

RELIANCE ELECTRIC COMPANY
CLEVELAND, OHIO 44117 U.S.A.

CURVE NO. 609347-22

PERFORMANCE STANDARDS FOR LONGWALL POWER SYSTEMS

Generally there are three criteria to be considered when evaluating the predicted operational performance of a longwall power system. These are as follows:

a) Voltage at motor terminals when all the plant is operating under full load conditions.

b) Voltage at the motor terminals when a major load (such as the A.F.C.) is being started.

c) Voltage at the motor controls under the conditions stated in (b)

a) The voltage at the motor terminals under full load conditions can usually be maintained at 95 - 98% of rated motor voltage simply by suitable choice of transformer voltage tappings. Some of this compensation is already inherent in the standards commonly used in the mining industry. (In the U.K. NCB standard motors are required to produce their specified performance at 1050 V. rather than the nominal 1100 V. of the supply. In the U.S.A. it is customary to use a 950 V. rated motor on a nominal 1040 V. supply.)

b) In recent years there has been a significant increase in the size of individual motors used in longwall applications and also in the total amount of power applied to single items of plant. Shearers are commonly up to 500 HP. and AFCs often have two or three motors each being rated at up to 300 or 400 HP. The large starting currents drawn by these motors can cause excessive voltage drops, and since the torque produced by an induction motor is approximately proportional to the square of the terminal voltage, the performance of these motors can be severely restricted.

It is very important to ensure that the performance characteristics of high horsepower motors are suitable for longwall applications. Available motors usually have a "locked rotor" currents of at least six times full load current, but some manufacturers have supplied motors having much higher values of "locked rotor" current which are quite unsuitable for a longwall application.

The nominal "pull-out" torque of a suitable A.F.C. motor should not be less than 2.6 x Full Load Torque at rated voltages. This means that the actual pull-out torque produced for various reductions in terminal voltage will be approximately as shown in the table.

% Rated Terminal volts	60	70	80	90	100
% Full Load Torque	0.94	1.27	1.66	2.11	2.60

Sizing of transformers for longwall installations

It has been common practice to size transformers according to the aggregate of installed motor horsepower. Typically an allowance of one kVA of transformer capacity per H.P. of motor load was considered very adequate. In such cases one relies upon the inherent overload capacity of transformers to supply the starting currents.

In recent years the H.P. of individual motors used on longwall installations has increased, quite dramatically. Where these large motors are supplied from comparitively small transformers the "one kVA per H.P." rule no longer provides an adequate supply for the starting currents.

Two simplified examples serve to illustrate the point.

a) Five, 100 H.P. motors supplied at 1,100 V. from one 500 kVA transformer having 5% impedance.
Assume four motors running at full load current 4 x 50A = 200 A.
One motor starting 6 x 50A = 300 A.
Total current 500A Voltage drop 10%

b) One 500 H.P. motor supplied at 1,100 V. from one 500 kVA transformer having 5% impedance.
Starting current = 6 x 250A = 1500A
Voltage drop = 30%

Thus unless the voltage at the motor terminals is maintained at 70% of rated voltage the starting torque will be less than full load torque. This would be clearly an unsatisfactory situation because the torque required to start a loaded AFC is considerably more than the running torque. The pull out torque (which is usually also the maximum torque) is developed at about 92% of synchronous speed when the current maybe 3 to 4 times full load current. Correct filling adjustment of the fluid couplings results in a motor current of 2.25 to 2.75 x FLC when the conveyor is stalled. Hence for a two motor AFC drive the most critical condition occurs at the instant when the second motor is switched on whilst the first motor is driving a stalled fluid coupling.

c) In longwall power systems most of the voltage drop occurs on the supply side of the motor control gear. This means that the control circuits are subjected to substantial voltage regulation and in some cases this voltage regulation may be sufficient to cause maloperation of the control circuits. The most severe condition usually occurs at the switch on the largest motor, (or alternatively in the event of two motors being switched on simultaneously). The high locked rotor starting current causes a voltage drop throughout the longwall electrical system. It is important to ensure that the voltage at the motor controls never falls to a value below which the controls will malfunction. Most manufacturers design their mining motor control equipment to operate down to 75% nominal voltage.

COMPUTATION

The program used is an adaptation of a program developed by Pennsylvania State University under a U. S. Bureau of Mines contract. The program was developed for the analysis of interactive mine power systems.

The single line network shown on Sheet #1 was used as the basis for the present calculations. The values of the impedance elements shown were varied in successive stages to simulate the operational conditions described. Eleven different conditions were computed.

Sheet #2 shows the transformer and cable impedance values used when referred to a common base voltage of 1,100 V. line to line.

Sheet #3 gives all the impedance values used with element connection numbers in an input format recognised by the computer program.

Sheet #4 is a precis of the computed voltage under the eleven sets of conditions indicated. The voltages are expressed as a percentage of the nominal 1100 V. for each condition two sets of voltages are given. The first line assumes no voltage compensation by tap changing and the second line assumes 10% overall compensation by tap changing.

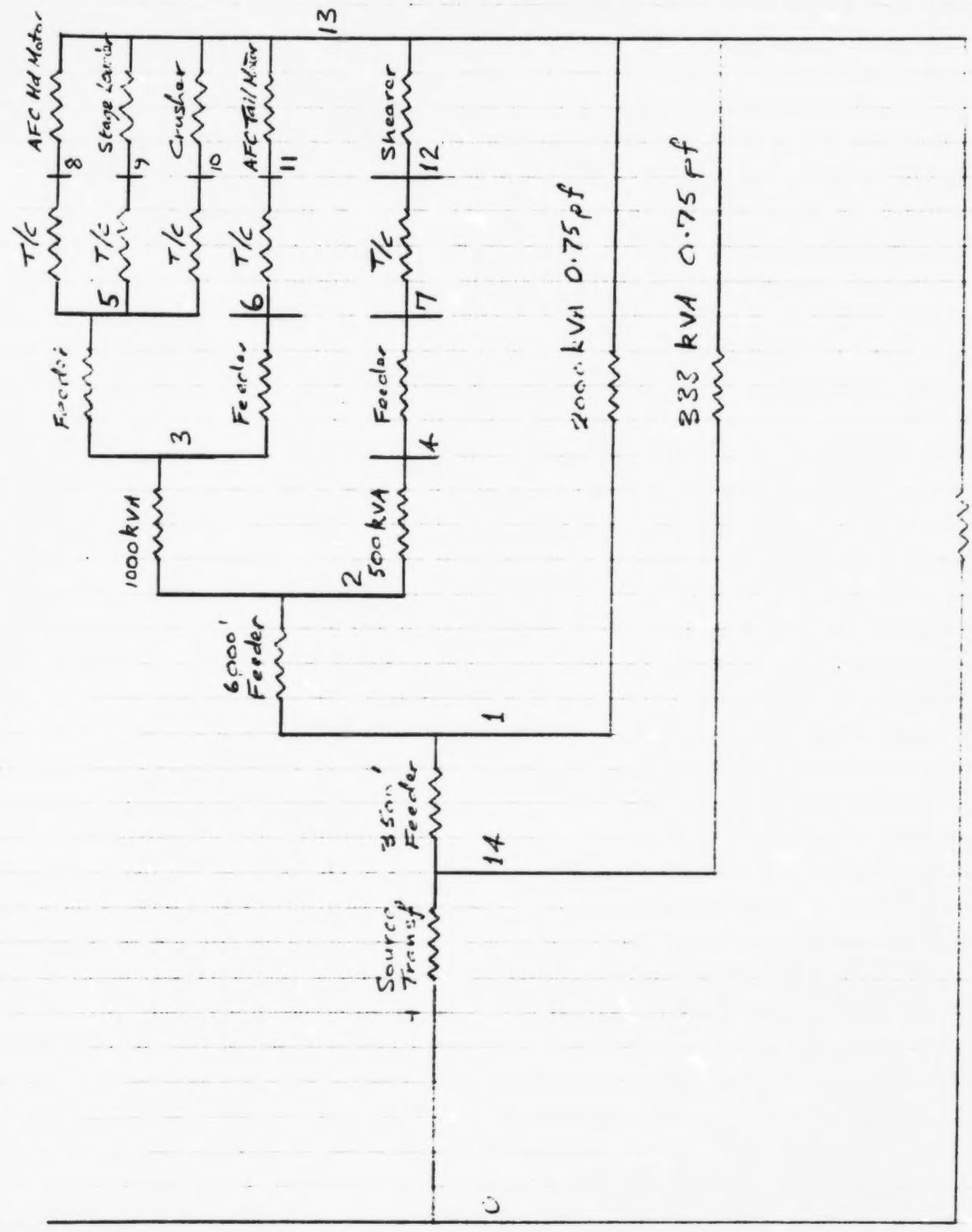
SERVICE MACHINE

INNOVATION IN PRODUCT DESIGN

DATE Nov 1982

SHEET NO. 1 SMOKY RIVER

SERVICE MACHINE CO.
6072 OHIO RIVER RD. HUNTINGTON, WV 25702 / P.O. BOX 8177 HUNTINGTON, WV 25705 PHONE 304/736-8933 TWX 710 931 1928



#9 Condition - Criterion (b)

(Data as Condition #8)
Under starting conditions, and with 10% voltage boost, the voltage at the Shearer Motor terminals is increased from 69.4% to 75.0% with the 1,000 kVA transformer.

#9 Condition - Criterion (c)

(Data as Condition #8)
Under starting conditions, and with 10% voltage boost, the voltage at the Shearer Motor Control is increased from 75.1% to 81.1%. This is considered to be an improvement which is essential to ensure correct operation of typical mining motor control gear.

#10 Condition - Criterion (b)

(Data as Condition #9, but Shearer trailing cable reduced from 3/0 to 2/0).
With 10% voltage boost and Shearer Motor starting, terminal voltage is reduced from 75.0% to 74.3%. Although this is an adverse change the magnitude of the change is insignificant compared with other benefits.

#10 Condition - Criterion (c)

(Data as above)
Under the above conditions the voltage at the Shearer Motor Control is increased from 81.1% to 81.3% - an insignificant improvement.

#11 Condition - Criterion (a)

(Data as above)
With 10% voltage boost and all plant at full load the voltage at the Shearer Motor terminals is:
With 500 kVA transformer and 3/0 cable - 94.5%
With 1000 kVA transformer and 3/0 cable - 95.5%
With 1000 kVA transformer and 2/0 cable - 95.0%
The last combination is satisfactory.

SERVICE MACHINE

INNOVATION IN PRODUCT DESIGN

DATE NOV. 1982

SHEET NO. 2 SMOKY RIVER

SERVICE MACHINE CO.
8072 OHIO RIVER RD. HUNTINGTON, WV 25702 / P.O. BOX 8177 HUNTINGTON, WV 25705 PHONE 304/736-8933 TWX 710 931 1928

SYMBOL	PLANT ITEM DESCRIPTION	CABLE LENGTH	COND. SIZE	CABLE TYPE	ACTUAL IMPEDANCE	IMPEDANCE AT 1,100V
	TRANSFORMER + SOURCE IMP.					0.0274/75°
		3,500'	3/0	8kV grade feeder	0.301/22°	0.0863/22°
		6,000'	3/0	"	0.516/22°	0.0143/22°
				"		
	TRANSFORMER 1,000KVA 6.6kV/1100V				6%	0.0726/80°
	TRANSFORMER 500 KVA 6.6kV/1100V				5%	0.1210/80°
	1100V FEEDERS	150'	3/0	2 kV grade feeder	0.0092/19°	0.0092/19°
	SHEARER TC	650'	3/0	"	0.0598/19°	0.0598/19°
	AFC HD TC	50'	2/0	"	0.0056/16°	0.0056/16°
	AFC TAIL TC	650'	2/0	"	0.0728/16°	0.0728/16°
	STAGE LDR TC	50'	#1	"	0.0088/11°	0.0088/11°
	CRUSHER TC	50'	#1	"	0.0088/11°	0.0088/11°
	AFC TAIL TC	650'	4/0	"	0.0728/13°	0.0728/13°

SERVICE MACHINE

INNOVATION IN PRODUCT DESIGN

DATE Nov 1982

SHEET NO. 3 SMOKY RIVER

SERVICE MACHINE CO.
6072 OHIO RIVER RD. HUNTINGTON, WV 25702 / P.O. BOX 8177 HUNTINGTON, WV 25705 PHONE 304/736-8933 TWX 710 931 1928

Number of elements = 21
Number of buses = 14

			Z	Φ
Source + Transf	0	14	2.74 E-002	7.500 E+001
3,500' 6.6kV Feeder	14	1	6.63 E-003	2.200 E+001
6,000' 6.6kV Feeder	1	2	1.43 E-002	2.200 E+001
1000 kVA Transf	2	3	7.26 E-002	8.000 E+001
150' 3/0 Feeder	3	5	9.20 E-003	1.900 E+001
50' #1 AFC T/C	5	8	5.6 E-003	1.6 E+001
250 HP AFC Motor	8	13	5.2 2 E+000	2.800 E+001
50' #1 S/L T/C	5	9	8.8 E-	1.1 E+001
150 HP S/L Motor	9	13	8.82 0 E+000	2.800 E+001
50' #1 Crusher T/C	5	10	8.9 E-003	1.100 E+001
150 HP Crusher Motor	10	13	8.82 0 E+000	2.800 E+001
150' 3/0 Feeder	3	6	9.20 0 E-003	1.900 E+001
450' 2/0 AFC T/C	6	11	7.2 E-003	1.600 E+001
250 HP AFC Motor	11	13	5.2 2 E-	2. E+001
2000 kVA at 0.75pf	1	13	5.51 0E-001	4.1 E+001
333 kVA at 0.75pf	14	13	3.306 E+000	4.1 E+001
PROGRAM LOOP	0	12	9.99 0 E+000	0.000 E+000
500 kVA Transf	2	4	1.21 E-001	8.0 E+001
150' 3/0 Feeder	4	7	9.200 E-003	1.900 E+001
450' 2/0 Cleaner T/C	7	12	5.980 E-002	1.900 E+001
* 500 HP Cleaner Motor	12	13	4.41 E-001	7.5 E+001
* 250 HP AFC Motor			7.36 0 E-001	7.5 E+001
500 HP Cleaner Motor			2.646 E+000	2.800 E+001
† 250 HP AFC Motor			2.117 E+	4. E+001
650' 2/0 T/C (AFC #1)			4.81 E-	E+001

* LOCKED ROTOR IMPEDANCE † 2[illegible], FLC

SERVICE MACHINE

INNOVATION IN PRODUCT DESIGN

DATE NOV 1982

SHEET NO 4 SMOKY RIVER

SERVICE MACHINE CO.
172 OHIO RIVER RD. HUNTINGTON, WV 25702 / P.O. BOX 8177 HUNTINGTON, WV 25705 PHONE 304/736-8933 TWX 710 931 1928

#	Condition	Variation	Voltage Ratio Assumed %	MotorControl AFC HD Motor 5	AFC HD Motor 8	Motor Control AFC Tail Motor 6	AFC Tail Motor 11	Motor Control Shearer Motor 7	Shearer Motor 12	S/L & Crusher Motor 9-10
1	Shearer starting. Other plant full load		100	83·6	83·5	83·7	82·6	68·3	63·1	83·5
			110	92·0	91·9	92·1	90·9	75·1	69·4	91·9
2	All plant full load.		100	87·8	87·7	88·0	86·8	87·8	85·9	87·7
			110	96·6	96·5	96·8	95·5	96·6	94·5	96·5
3	AFC HD Motor LR Other plant full load		100	77·6	77·3	78·2	77·1	84·7	82·8	77·5
			110	85·4	85·0	86·0	84·8	93·2	91·1	85·2
4	Both AFC Motors LR Other plant full load		100	70·1	69·8	70·2	66·6	82·2	80·4	79·0
			110	77·1	74·8	77·2	73·3	90·4	88·4	77·0
5	AFC HD Motor LR. Tail Motor 2.5 FLC. Other FL		100	75·8	75·5	76·1	73·8	84·0	82·1	75·7
			110	83·4	83·0	83·7	81·2	92·4	90·3	83·3
6	Both AFC Motors LR Other plant full load	T/C to AFC Tail Motor now 4/0	100	70·0	69·7	70·1	67·3	82·2	80·4	69·9
			110	77·0	76·7	77·1	74·0	90·4	88·4	76·9
7	All plant full load	(as 6)	100	87·8	87·7	88·0	87·2	97·8	85·9	87·7
			110	96·6	96·5	96·8	95·9	96·6	94·5	96·5
8	All plant full load	(as 16) + 1000 KVA Transf. in Shearer	100	87·8	87·7	88·0	87·2	98·9	86·8	87·7
			110	96·6	96·5	96·8	95·9	97·8	95·5	96·5
9	Shearer starting Other plant full load	(as 9)	100	83·1	83·0	83·3	82·5	73·7	68·2	83·0
			110	91·4	91·3	91·6	90·7	81·1	75·0	91·3
10	Shearer starting Other plant full load	(as 9) but Shearer T/C now R/O	100	83·1	83·0	83·3	82·5	73·9	67·5	83·0
			110	91·4	91·3	91·6	90·7	84·3	74·3	91·3
11	All plant full load	(as 10)	100	87·8	87·7	88·0	87·2	88·8	86·4	87·7
			110	96·6	96·5	96·8	95·9	97·7	95·0	96·5

ANALYSIS OF RESULTS

#2 Condition - Criterion (a)

(This corresponds with the data as supplied by McIntyre. All long-wall plant at full load)
With 10% voltage boost on tap settings the motor terminal voltages are 95.5% for the AFC Tail Motor and 94.5% for the Shearer. This is considered acceptable.

#1 Condition - Criterion (b)

(Data as supplied by McIntyre) Shearer starting - all other plant full load.
With 10% voltage boost the Shearer terminal voltage is 69.4%. This is acceptable for a motor starting without any substantial mechanical load.

#1 Condition - Criterion (c)

(Data as above)
Even with 10% voltage boost, the voltage at the Shearer Motor Control is down to 75.1%. This is considered to be too low to guarantee correct operation of the control circuits.
Most of the voltage drop is due to the 500 kVA transformer (See Conditions 8 to 11).

#3 Condition - Criterion (c)

(Data as above)
With 10% voltage boost the voltage at the AFC motor control will not fall below 85.4% when the AFC Head Motor is started. This is satisfactory.

#3 Condition - Criterion (b)

(Data as above)
With 10% voltage boost the voltage at the AFC Head Motor terminals will not fall below 85% when the motor is started. This will produce a locked rotor torque of 1.86 x FLT which is satisfactory.

#4 Condition - Criterion (c)

(Data as above)
With 10% voltage boost and both AFC Motors starting simultaneously the lowest Motor Control voltage is 77.1%. Bearing in mind that this condition should not normally occur the figure is satisfactory.

#4 Condition - Criterion (b)

(Data as above)
With 10% voltage boost and both AFC Motors starting simultaneously the AFC Tail Motor has the lower terminal voltage of 73.3%. Bearing in mind that this condition should not normally occur the figure is satisfactory.

#5 Condition - Criterion (b)

(Data as above)
AFC Head Motor at switch-on, AFC Tail Motor assumed driving stalled fluid coupling with Tail Motor taking 2.5 x FLC. (This is the most critical condition which should normally occur if the fluid couplings and the motor control gear are correctly set up.) With 10% voltage boost the AFC Tail Motor has a terminal voltage of 81.2% corresponding to 1.43 x FLT. This is considered to be satisfactory on the basis of comparison with other installations.

#5 Condition - Criterion (c)

(Data as above)
With 10% voltage boost the lowest AFC Motor Control voltage is 83.7%. This is satisfactory.

#6 Condition - Criterion (b)

This condition shows the effect of increasing the AFC Tail Motor trailing cable from size 2/0 to size 4/0. Comparison between conditions #4 and #6 show that the voltage at the AFC Tail Motor terminals is increased from 73.3% to 74.0%. This is not considered to be a worth while improvement.

#7 Condition - Criterion (a)

(Data as above, AFC Tail Motor trailing cable size 4/0, all plant full load current.)
Comparison between conditions #2 and #7 shows that under full load, with 10% voltage boost, the voltage at the AFC Tail Motor terminals is increased from 95.5% to 95.9% by increasing the trailing cable size from 2/0 to 4/0. This is not considered to be a worthwhile improvement.

#8 Condition - Criterion (a)

(Data as above, with transformer supplying Shearer increased from 500 kVA to 1,000 kVA)
With 10% voltage boost the full load terminal voltage at the Shearer Motor is improved from 94.5% to 95.5%. This is not in itself a significant improvement.

APPENDIX 4-3

LONGWALL FACE TRAINING PROGRAM
ASSOCIATED MINING CONSULTANTS LTD.

LONGWALL FACE TRAINING PROGRAM

PREPARED FOR

McINTYRE MINES LIMITED

SUBMITTED BY

ASSOCIATED MINING CONSULTANTS LTD

DECEMBER 1982

AR-57.1

LONGWALL FACE TRAINING PROGRAM

1 Introduction

A high output longwall face installation represents a major capital investment on which a satisfactory return is essential. This can only be achieved if all personnel involved in the provisioning, operation, and servicing of the longwall face are properly trained. Such training, whilst emphasizing the skill requirements of each individual task or operation, must also encompass the interdependence of those skills so that the collective efforts of management and labour are fully co-ordinated to the attainment of the objective of a highly productive and safe longwall working unit.

Associated Mining Consultants Ltd. (AMCL) in association with their parent company British Mining Consultants Limited (BMCL), have developed an effective program to enable this objective to be attained efficiently, effectively and economically within the shortest possible time.

2. Approach

AMCL would assemble a training team consisting of three key longwall personnel in the following categories:

- A mining instructor/face mechanization engineer
- An advanced mechanical instructor
- An advanced electrical instructor

One of the foregoing would act as Team Leader. The team, which would include NCB seconded personnel, would be assisted by support

personnel in preparing the training material. The work would be done under the overall direction of a senior staff member who would act as Project Manager and initiate the work and monitor the program.

Development of the training program would commence with an assessment of the training required. This would include:

- Identification of the tasks to be performed
- Determination of skill level of the labour force
- Preparation of detailed job descriptions for determined operational tasks

This would be carried very early in Phase I by the Project Manager and Team Leader, and would take about 5 days. Phase I would continue with preparation of course material, taking the 3-man team ten days to prepare.

Phase 3, actual implementation of the training program, surface training installation and commissioning and operational assistance would take approximately 10 weeks, broken down as follows:

- Surface Mock-up - 3 days
- Surface Training - 20 days
- Installation/Training - 10 days
- Operational Assistance - 15 days

To make the necessary preparations for a smooth start to the work the Team Leader accompanied by the Project Manager would precede the remainder of the team by one week.

3. Key Features of the Program

The key features of the program are UK based training and on site training.

UK Based Training

Exposure of key staff from the mine to existing longwall installations in the UK.

On Site Training

On site training of all categories of personnel to be involved in longwall operations, namely:

- Underground Manager
- Underground Foreman
- Longwall Coordinator/Overman
- Mine Engineer
- Mine Electrician
- Mine Training Officer
- Face Mechanics
- Face Electricians
- Face Crews (including Stage Loader Operator)
- Machine Operators

The on site training program will include:

- Introductory material of a general and informative nature applicable to a wide level of mine personnel.

- Assistance with the erection on the surface of a working section of the face equipment.

- Hands-on practical instruction based on the surface installation of the actual equipment, with manufacturers' representatives in attendance, together with an adequate classroom facility for lectures and discussions.

The main phases of the on site training program will include:

Phase 1 Preparation of program material.

Phase 2 Exposure of key staff to existing longwall operations.

Phase 3 Implementation of training at the mine.

The following additional phases are advised where the longwall installation is expected to be the first of several:

Phase 4 Post-course review and modification of material.

Phase 5 Evaluation and follow-up.

4. Training Program

Phase 1 - Preparation of Training Program Material

Development of the training program would commence with an assessment of the training required. This would be carried very early in Phase 1 by the Project Manager and Team Leader and would include:

- Identification of the tasks to be performed
- Determination of skill level of the labour force
- Preparation of detailed job descriptions for determined operational tasks

The program will be based on available NCB training material supplemented by equipment manufacturers' instructional material and data assembled during the onsite assessment period.

Preparation of the course outline involves:

- Glossary of longwall mining terms
- Introductory material on longwall mining
- Equipment identification
- Familiarization material relating to key component items
- Job descriptions including:
  - objectives
  - duties and responsibilities
  - performance criteria
- Practical advice on longwall operation and trouble-shooting, such as fault analysis, location and rectification on specific equipment items and system faults, such as horizon control, face alignment and conveyor creep

Based on the above outlines, detailed training programs and modular packages will be prepared which will include:

- Instructors' notes
- Course material
- Hand-outs
- Training aids

The mode of presentation of the training material is selected from films, 25 mm slides, transparencies for overhead projectors, diagrams, printed hand-outs and video-tape material as appropriate.

Phase 2 - Exposure of Key Staff to Existing Longwall Operations

Phase 2 should commence as early as possible and involve the following personnel:

- Line management staff responsible for the day-to-day running of the face, ie: Undermanager and Face Supervisor.

- Maintenance Foreman.

- Mine Training Officer.

This UK based program should be of at least 2-week duration and cover the following aspects:

- Theory and practice of longwall mining, organization of longwall systems. Mechanized aspects of longwall equipment supplied.
- Strata control - control of hazards on longwall faces.
- Visits to mines with relevant types of equipment.
- Visits to equipment manufacturers.
- View various stages of compatability trials, including a short training program on each item of equipment.

Phase 3 - Implementation of Training at the Mine

Phase 3 is associated with the arrival of the longwall equipment at the mine and applies to personnel on all the shifts selected to operate the equipment. It is scheduled so as to maximize the availability of equipment and includes an introductory program covering aspects such as:

- Sequence of operations
- The importance of good horizon control
- Types of delays and their effects
- Conveyor problems such as creep and over-loading
- Strata control
- Safety procedures
- Maintaining a straight face-line
- Dust suppression, etc.

The introductory program would involve:

- A seminar for senior mine management and section officials on the 'management of a longwall face'. A typical seminar outline is enclosed for information as Appendix A.

- 'Face-Team-Conference' type of seminar, involving the personnel indicated in Phase I plus senior mine management.

- Inspection of the equipment and a visit to the new face to observe practical aspects relating to the new face.

Training of Face Crews in the Installation and Operation of Equipment

Based on the prepared training packages face crews would be trained on a team-concept to become competent to carry out all tasks. The period involved would be influenced to a considerable extent by the ability of the people selected to assimilate the package. A ten-day integrated course for supervisors, face team and mechanics involving some five days' theory (Figure 1) with the balance as practical instruction is envisaged. A practical pre-test and post-test element would be included. Additional instruction is usually desirable for the machine operators and would involve an additional five days.

After instruction, the recently trained face crews would be involved in dismantling the equipment, transporting it underground and installing it on the face. After installation and commissioning of the face equipment, the program continues with hands-on instruction as the face moves off and commences production.

Training of Maintenance Men on Installation, Servicing and Maintenance.

This involves a close association with manufacturers' representatives and, as necessary, arrangement of tests with their collaboration to assess the understanding of the trainees and test their competency. Here, also, the period of time is dependent on the background and experience of the individuals concerned.

If higher-skilled mechanics from conventional mining sections can be recruited for the new face, an initial 15-day program, consisting of five days' theory (Figure 1) and ten days' practical instruction, should suffice. The package would include familiarization with all aspects of the equipment and incorporate fault-finding procedures and maintenance check lists. Understanding of the subject would be tested on a stage-by-stage basis.

Training of Remaining Mine Training Staff.

In addition to the training officer and chief instructor, selected for the new face, familiarization with longwall mining techniques would be given to other training staff at the mine. This involves assisting the chief instructor to select and train instructors to assist with any subsequent re-training of replacements, and staff for subsequent longwall installations. If practicable (insofar as these people would have duties in other parts of the mine) they should be associated with the various stages of the training program. A final five-day course in instructor training would then be given.

Phase 4 - Post-Course Review and Modification of Material

Phase 4 takes place at the end of the overall training program and is designed to improve subsequent training programs. The review is based

on relevant information relating to training modules gathered at the end of each individual training session.

The objective is to review the appropriateness of the overall program and each material package, subsequently determining through discussion with the people involved whether minor modifications would be desirable for subsequent installations. It is usual for such modifications to be effected by the company and mine training staff with AMCL staff acting in an advisory role.

Phase 5 - Evaluation and Follow-up

Phase 5 should take place some three months after the initial training program has been completed and the equipment installed. It involves a one-week visit to the mine to review progress and have discussions with mine training staff and corporate executives.

Irrespective of how successful the initial training appears to have been, Phase 5 is considered vital if the company is to successfully apply the program to its other installations.

5. Facilities Required for Training Program

Surface Mock-up

Conveyor:	Short length with tail and head drive arrangement. 18 pan lengths.
Shearer:	Mounted on conveyor: Powered - ability to haul shearer along short length of pan line would be an advantage.
Supports:	4 shields in position preferably at head end should be prepared.

Sivad: Set up in suitable classroom area.
Demonstration console and suitable number of keys, etc.
Coupled up, operational.

Cover: Some form of cover over part or all of surface mock-up.

Classroom

Accommodation: Approximately 20 people

Equipment:
Overhead projector
16 mm film projector
35 mm slide projector
Screen
Blackboard
Sivad demonstration console

16 mm films:
Winning the Coal
Powered Supports
Pressure and Flow
Safety in Longwall Mining

35 mm slides:
Longwall Installation Set
Fluid Couplings
Shearer Circuit
Shield Circuit
Sivad Mark 2

6. Cost Estimate

		Mandays	Fees $	Expenses $	Total $
Phase 1	Preparation	45	26 500	8 425	34 925
Phase 2	Exposure	10	15 000	600	15 600
Phase 3	Training	145	83 000	21 325	104 325
Phase 4	Review	15	8 500	1 525	10 025
Phase 5	Evaluation	10	6 500	4 400	10 900
	Total	225	139 500	36 275	175 775

Fees include all professional charges associated with course preparation, training program, review and evaluation, including charges associated with the visit of senior mine staff to existing UK longwall operations.

Expenses include all airfares, accommodation and living expenses, ground transportation, preparation of training aids, communications and copying. Should the mine be in a position to provide such items as accommodation and ground transportation, these expenses will be reduced accordingly. Air travel and living expenses associated with the visit of senior mining staff to existing UK longwall operations are not included.

FIGURE 1

LONGWALL FACE TRAINING PROGRAM

McINTYRE MINE LONGWALL INSTALLATION

SUBJECT - POWERED ROOF SUPPORT SYSTEM

DAY	Session 1	Session 2	Session 3	Session 4
1	Introduction to P.R.S. System. Outline Course Programme Pre-Test, Film on P.R.S. Function of P.R.S.	Function of P.R.S. Cont'd. Setting Force/Yield Force Types of Supports. Chocks, Chock Shield, and Shields Operate Support System.	Use of Hydraulics for P.R.S. Basic Principles. Explanation of Symbols used in P.R.S. Circuit Diagrams. Types of Circuits and Valves.	Film:— Pressure and Flow Questions and Answers on previous work.
2	Power-Pack:— General Layout. Identification of Components. Details of Tanks and Fittings.	Power-Pack:— Pumps. Type and Specification. Design of Drive, Lub. System, Valve Chest and Safety Devices.	Power-Pack:— Valve Gear. Explanation of Hyd. Circuit. Operation of Relief Valves, Onloader Valves, Reducing Valves etc.	Schedule and procedure for maintenance. Practical Work:— Identify and examine all components on Power-Pack. Go through operational check.
3	Electrical details of Pump. Electrical Protection.	Electrical details of Tank. Electrical Protection.	Shield Support Layout of Shield, Identify all components. Explanation of control terms.	Details of legs, Reverse Thrust Ram and Ancillary Equipment.
4	Hydraulic Circuit:— Face Hose Layout. Use of Isolators and Relief Valves. Explanation of each Pressure Service.	Use of Pilot Operation. Multi-Bore Hoses. Explanation of each operational sequence in circuit.	Valve Pack Details of each module and valve and its operation.	Manifold Block. Shuttle valve manifold. Maintenance and Fault location procedure.
5	Practical work on Shields. Relate all information from drawings to supports	SPLIT INTO 2 GROUPS Group 1. To dismantle and examine spare modules. Group 2. Fault location on support system. GROUPS TO CHANGE ROUND		Post Test. Course Review and Open Forum.

FIGURE 1

APPENDIX A

A TWO DAY MECHANIZATION COURSE FOR OFFICIALS

A TWO DAY MECHANIZATION COURSE FOR OFFICIALS

The course is designed to give Officials and Supervisors a working knowledge of fully mechanized longwall faces. Sessions must emphasize the supervisory aspects of operations, and course members should be given practical instruction in the respective operations. Safe working procedures in line with any Mining Regulations should be highlighted and adopted at all times.

This course would span two full days of four sessions each. There is no time allowed for introductions, administration details or pre and post tests.

Session I Principles of Longwall Mining

Basic colliery layout of:

- Roadways
- Ventilation
- Development
- Direction of travel of face

Discuss advantages or disadvantages of retreat and advancing longwall faces:

- Delays due to face end work
- Possible elimination of roadway work
- Previously driven roadways prove the seam and can identify any faults in the strata
- When face is established need for supplies is reduced
- Efficiency of ventilation
- Salvaging equipment where face is finished

Session 2 Design and Method of Operation

Course members should be able to identify apparatus by name and a Training Face installation should be viewed. During this session consideration should be given to:

- The types of equipment
- The installation
- The method of operation

Basic Equipment and Type

- Roadway Conveyors, Stage Loaders and Armoured Face Conveyors
- Power Loaders
- Roof Supports
- Anciliary equipment, power packs, pumps etc.

Planning the Installation

Consideration should be given to:

- Roadways en route to the face
- Methods of transport and handling
- Sequence of needs of the equipment
- Siting of ancilliary equipment
- Communication system
- Electrical supply

Session 3 and 4 Equipment and Type

Build up a complete mechanized face on a board using magnetic aids to show:

(1) The Stage Loader

Discuss:

- Length of cover over gate conveyor
- Types of pans
- Anchoring
- Drive

(2) The Armoured Face Conveyor

Discuss:

- Pans
- Ramp plates
- Captivating plates
- Spill plates
- Cable Brackets and handling
- Special bolts
- Tensioning
- Anchoring
- Drive

(3) The Power Loader

Discuss:

- Under frame
- Guides
- Drum and cutting picks
- Gear head
- Motor and Switch chamber
- Haulage unit

(4) Powered Supports

Discuss:

- Operation
- Basic hydraulic circuit and operation

(5) Anciliary Equipment

Discuss:

- Pumps
- Haulage or winches

Session 5 and 6 Electrical Aspects

Using recommended safety procedures and Mining Regulations as the key point, discuss:

- The Mine electrical distribution system
- Electrical distribution at the coal face to include:
    - Transformer position
    - Basic operation of gate end box
    - Signalling and communication systems
    - Lighting systems
    - Types of cable, main power supply and trailing cables
    - Flameproof and intrinsically safe apparatus
    - Protection

Session 7 Practical Operation of Equipment

Each course member should be allowed to operate:

- Switchgear
- Conveyor and controls
- Power loader
- Roof supports and snaking the conveyor
- Signals and communication systems

Session 8 Cutting Horizon Strata and Dust Control, and Maintenance

Discuss:

- Necessity to keep face in line, damage to AFC components and poor roof conditions

- Importance of horizon control maintaining uniform surfaces of roof and floor - extracting maximum amount of coal and minimum dirt
- Use of water for dust control - enough to be effective

Care and Maintenance Requirements

Checks to be carried out as follows:

Armoured Face Conveyor

- Chain correctly tensioned
- Missing flights
- Slack or missing connector bolts
- Condition of sprocket and bolts
- Chain strippers - effectiveness and security
- Chain guide wearing plate on drive headframes or ramp sections - excessive wear and grooving
- Sideplates in good condition and bayonets connecting cable troughs in position
- Ramp plates for security
- Fluid coupling, gearbox levels and leakage
- Drivehead and return anchorage facilities in good condition and being used
- Inspection pans for missing or loose staples of plate damage
- Delivery to stage loader
- Measurement of chain stretch
- Drive (Motor Gearbox and Cable)

The following should not be allowed:

- Conveyor running with unnecessary bends
- Conveyor running with slack chain
- Any other bolts than ones supplied used
- Use of bolts more than once
- Conveyor running with bent flights
- Bad delivery to stage loader
- Advancing face conveyor when chain is not running
- Advancing of conveyor too close behind power loader

Power Loader

- Pre-start warning effective

- Device securing drum to shaft
- Badly worn or broken pick boxes
- Sprays are effective and protected
- Cowl in good condition and doing its work
- Machine securely fastened to underframe
- Cable and hose towing equipment - must lay them centrally in trough
- Bollards and chain guards in position - bollards free to rotate
- Oil levels

Powered Supports

- Soluble oil/water mixture (5% soluble oil)
- Losses by leakage
    - Pump gland
    - Hose and connections
    - Faulty valves
    - Leg seals
- Tank filters and high pressure filters
- Hoses and nipples protected from damage
- Lubrication of pump
- Tank for sludge content

Discuss importance of lubrication on all items of plant.

Discuss availability of spares.

POOR

QUALITY

ORIGINAL